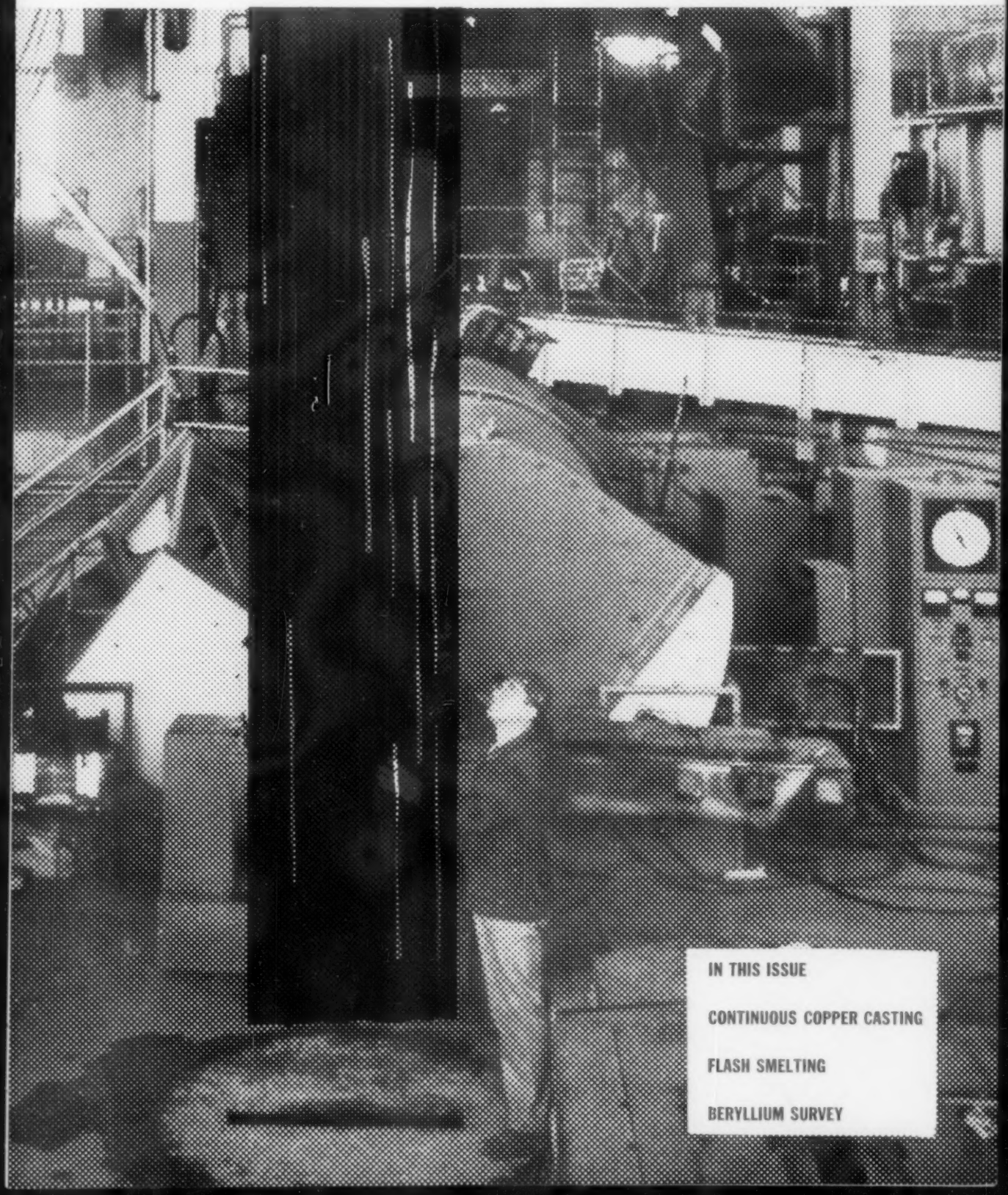
JOURNAL OF

METALS

JUNE 1958



IN THIS ISSUE

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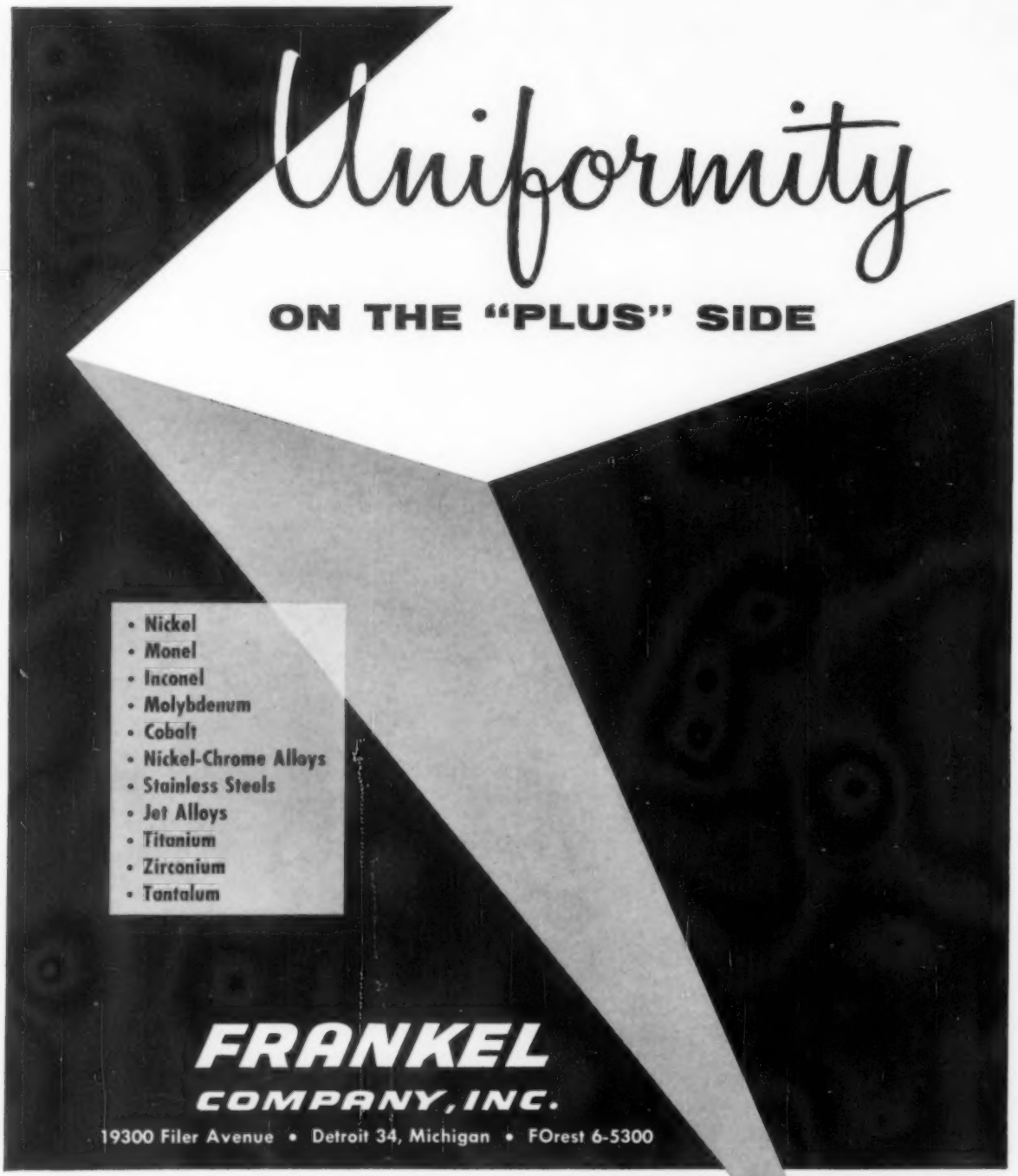

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JOURNAL OF

METALS

VOL. 10, NO. 6 JUNE 1958

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COVER

Continuous casting of tough-pitch copper, a "first" for the American Smelting and Refining Co., is the subject of this month's cover, designed by Naomi Lazard. The complete story of this new nonferrous development will be found on p. 404.

FEATURES

Personnel	376	Education News	382
Books	380	Personals	429
Reporter	387	Coming Events	434
Editorial	389	Advertisers Index	434
AIME News	425		

ARTICLES

Flash Smelting of Copper Concentrates — Petri Bryk, John Ryselin, Jorma Honkasalo and Rolf Malmstrom — 395

Beryllium: Present and Potential Uses — Lawrence F. Boland — 401

Tough-Pitch Copper Continuously Cast — F. Weston Starratt — 404

Smelting Titaniferous Ores — H. U. Ross — 407

Suspended Basic Roofs for Copper Furnaces — G. Bridgstock, and J. Towers and R. N. Brown — 412

Direct Electrorefining of Nickel Matte — L. S. Renzoni, R. C. McQuire, and W. V. Barker — 414

Iron and Nickel by Carbonyl Treatment — R. M. Lewis, J. W. Cookston, L. W. Coffer, and F. M. Stephens, Jr. — 419

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Plant Metallurgist, 30-40, with degree in metallurgy, metallurgical engineering or ore dressing and a minimum of five years experience in base metal sulfide plants and laboratory work. Under line control of principal plant superintendent and functional control of sr. metallurgist. Will conduct laboratory studies and be responsible for applications of improvements to plant circuits. Salary commensurate with experience; housing available; 33-month contract, transportation and other benefits. Location, South America. F5592(b).

BOOKS

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The Continuous Casting Of Steel, by M. S. Boichenko, V. S. Rutes, and N. A. Nikolayev, *Consultants Bureau Inc.*, 227 W. 17 St., New York 11, N.Y., 25 pp., $10.00—Originally published in Moscow, Russia, 1956, this book reviews the Russian achievement in solving this important problem. Covered are experimental work and commercial installations, including quality and economic cost.

(Continued on page 380)

Additional classified advertising appears on pages 380, 391.

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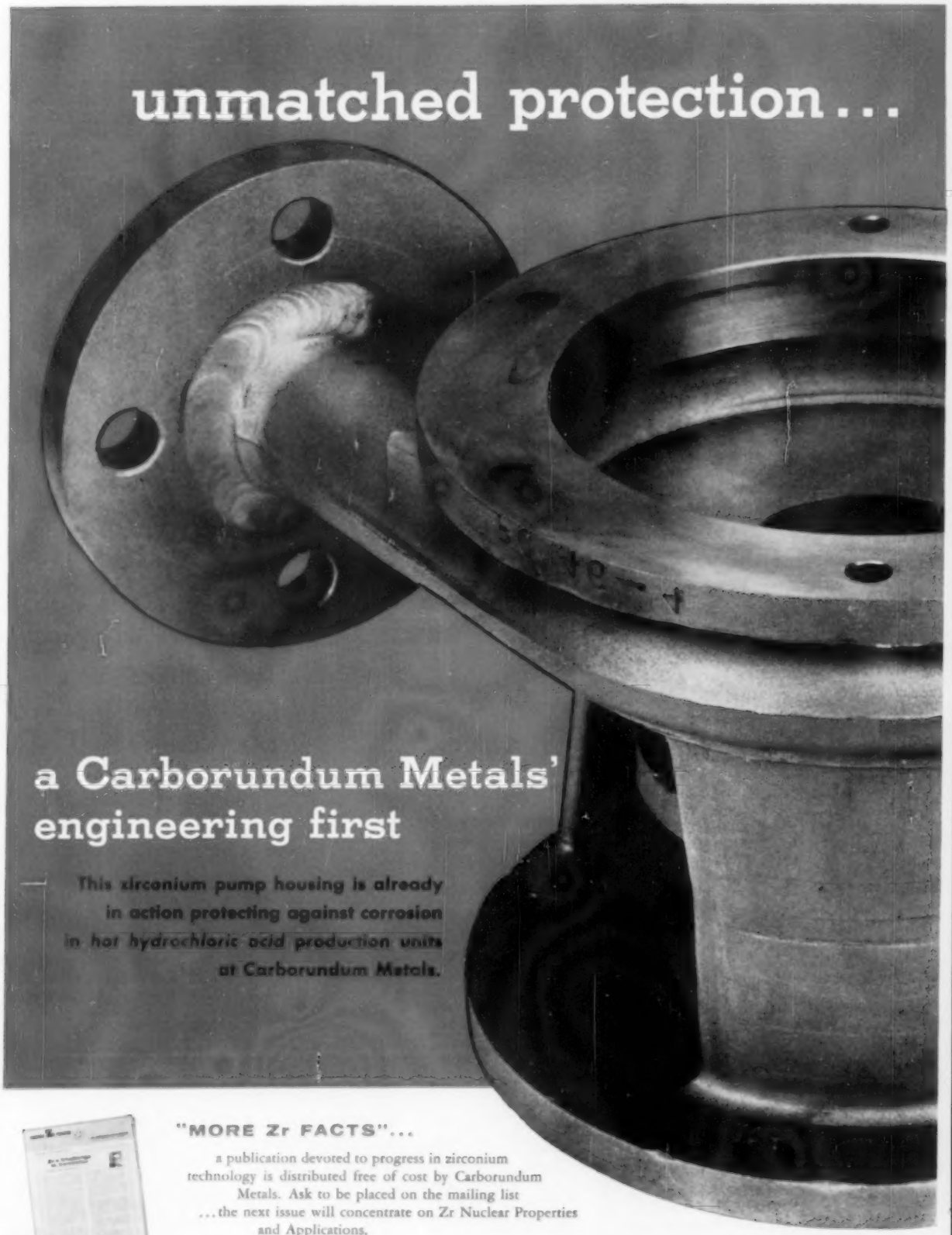

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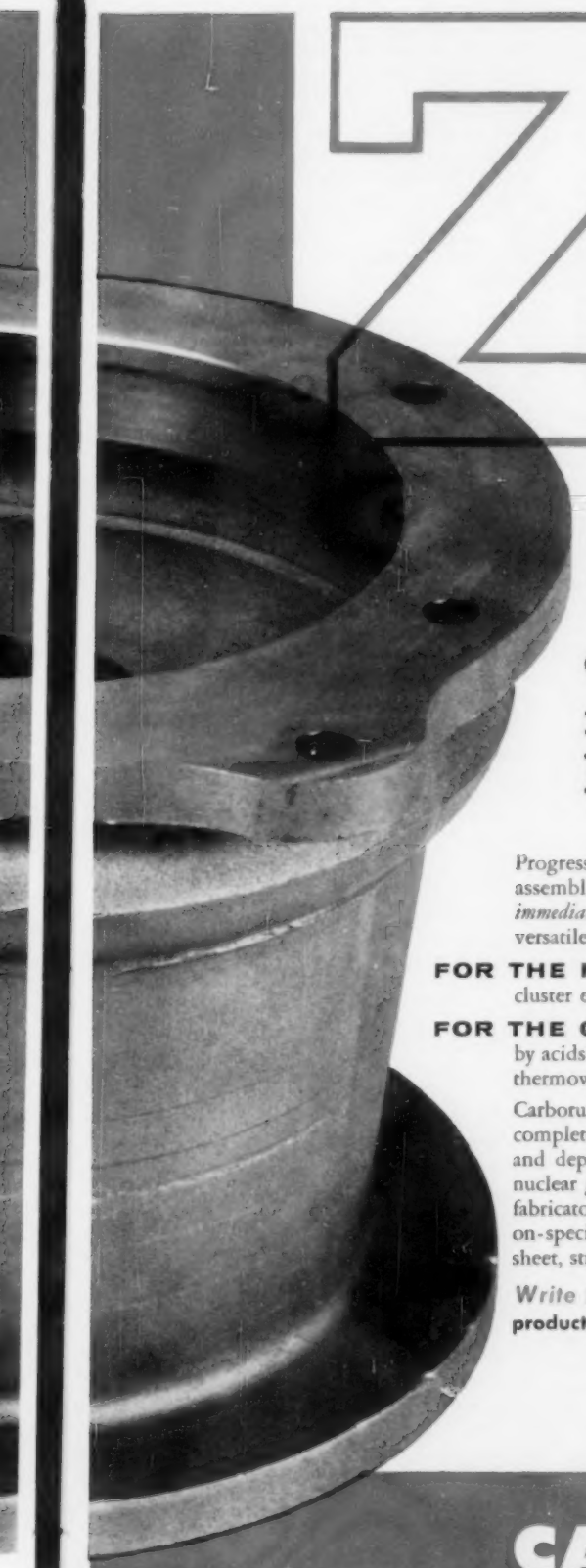

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Progress in zirconium fabrication techniques is evidenced in intricate cast-weld assemblies such as this impeller housing. Similar zirconium components are *immediately available* through Carborundum Metals for system designs where the versatile properties of this metal provide outstanding benefits:

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Books

(Continued from page 377)

Der Bau Und Der Betrieb Der Kupolöfen, *William Knapp-Verlag,* Haale (Saale) Germany, Second Edition, two volumes, 163 and 219 pp., $2.65 and DM 9.60, 1953–1956—A concise treatment of the design and the operation of cupola furnaces for the engineer or practical foundryman. Vol. I (1956) covers the development of the cupola furnace smelting technique, the types of furnaces, and auxiliary equipment. Vol. II (1953) covers raw materials and fuels, chemical and physical phenomena in furnace shafts, and the composition and measuring of the charge. Some of the 450 literature references are given.

Plant Design and Economics, by Max S. Peters, *McGraw-Hill Book Co. Inc.,* 511 pp., $11.00, 1958—The first part of this book deals with the applied economics, particularly in the process industries in design work. The various costs involved in industrial processes, capital investments, and investment returns, cost estimation, cost accounting, cost optimum economic design methods, and other subjects dealing with economics are covered both in qualitatively and quantitatively. The remainder of the book deals with methods and important factors in the design of plants and equipment. Generalized subjects, such as waste disposal, structural design, and equipment fabrication, are included along with design methods for different types of process equipment. Basic cost data and cost correlations are also presented for use in making cost estimates. Illustrative examples and sample problems are used extensively to illustrate the applications of the principles to practical situations. Problems at the end of most of the chapters give the reader a chance to test himself as to his understanding of the material. Several major design problems are included in the Appendix.

Power Reactor Technology, prepared by Dr. Walter H. Zinn and Assoc. of the General Nuclear Engineering Corp. under contract to the *U.S. Atomic Energy Commission,* Superintendent of Documents, U.S. Government Printing Office, Washington 25, D.C., 1958—This new journal in the Commission's Technical Progress

Behavior of Metals at Elevated Temperatures, *Philosophical Library,* 15 E. 40 St., New York 16, N. Y., 122 pp., $6.00, 1957.—This volume, illustrated with 63 diagrams and 3 pages of art plates, covers the effect of temperatures from only slightly above normal to the highest working limits. Among the information discussed is: the effects of high temperatures on the engineering properties of metals; the effect of temperatures up to 450°C, with particular reference to airframe structures and guided missiles; the effects of temperatures of 500°C upwards on nonferrous metals, high temperature steels, and kinetic heating. •

Dangerous Properties of Industrial Materials, by N. Irving Sax, *Rienhold Publishing Corp.,* 1467 pp., $22.50, 1957.—Over 8500 hazardous industrial materials, including trade-name products, are covered in this new edition of the book formerly called *Handbook of Dangerous Materials;* 1000 pp. are devoted to the alphabetical general chemical section, with nine smaller sections on toxicology, ventilation control, personnel protection and personal hygiene, atmospheric pollution, radiation hazards, industrial fire protection, storage and handling of hazardous materials (explosives, etc.), reactor safeguards, and allergic disease in industry. Detailed information is given on hazardous properties and situations, safety precautions, symptoms and treatments, shipping regulations, and the important characteristics of the individual materials. An extensive cross reference system provides ready access to the information contained in the volume. •

Grain Boundaries in Metals, Monographs on the Physics and Chemistry of Metals, by D. McLean, *Oxford University Press,* 114 Fifth Ave., New York 11, N. Y., 343 pp., $7.05.—This book is believed to be the first devoted to grain boundaries. Because the influence of grain boundaries is widespread, many topics are covered. Discussed are ideas about the structure of grain boundaries, their influence on microstructure, diffusion, fracture, and behaviour during plastic deformation. Other chapters deal with the energies of grain boundaries, segregation of impurities to grain boundaries, sliding at, and migration of, grain boundaries, and the special features of low angle boundaries. The author has attempted to clarify the basic problems involved, and a good deal of numerical data is given which should be useful for reference. A bibliography of several hundred publications is also included. •

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First, there had to be germanium of flawless structure and unprecedented purity. This was obtained by growing large single crystals—and creating the "zone refining" technique to purify them to one harmful part in *ten billion*.

The "junction" transistor, another radical advance, spurred transistor use. Easier to design, lower in noise, higher in gain and efficiency, it became the heart of the new electronics.

An ingenious technique for diffusing a microscopically thin layer on semiconductors was created. The resulting "diffused base" transistor, a versatile broadband amplifier, made possible the wide use of transistorized circuits in telephony, FM, TV, computers and missiles.

In telephony the transistor began its career in the Direct Distance Dialing system which sends called telephone numbers from one exchange to another.

For Bell System communications, the transistor has made possible advances which would have been impossible or impractical a brief decade ago.

1958—*Satellite transistor, incorporating 10 years of Bell Labs research and development.*

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News of EDUCATION

MATERIALS FOR NUCLEAR REACTORS

A special one-week course in *Materials for Nuclear Reactors* will be offered at **New York University** during the week of June 9-13, 1958. Sponsored by the Department of Metallurgical Engineering and the Office of Special Services to Business and Industry, the course is designed for people with metallurgical or general engineering background who are concerned with the design and operation of nuclear reactors.

A series of 20 lectures by experts in their fields will be presented. Topics covered are: *Critera for Materials Selection*, June 9; *Properties of Materials for Reactor Applications*, June 10; *Environmental Effects* (including Radiation Effects), June 11; *Fuel Materials and Systems*, June 12; *and General Problems in Reactor Component Design*, June 13.

Four Lectures will be given each day, and informal discussions will be held in the evening. The course will be given at the University Heights Campus, and dormitory space will be available for those desiring it.

Fees are: full week, $110; single day, $35; $2.50 per night. Priority will be given full week registrants, and registration will be limited.

GOVERNMENT—UNIVERSITY RELATIONSHIPS

Increased Federal support of basic research at colleges and universities were among the views recently expressed by the **National Science Foundation** in a report entitled, *Government—University Relationships in Federally Sponsored Scientific Research and Development*.

The Foundation is responsible, under an Executive Order issued by President Eisenhower in March 1954, for making recommendations designed to serve the needs of Federal agencies and, at the same time, safeguard the strength and independence of the Nation's institutions of higher learning.

Copies of the report, may be purchased from the *Superintendent of Documents, Government Printing Office, Washington 25, D. C.*, for 40 cents.

ENGINEERING EDUCATION IMPROVEMENTS

The Carnegie Corp. of New York has granted $106,600 to **Case Institute of Technology**, Cleveland, for a one-year program of study and action aimed at producing major improvements in engineering education.

Among the aims of the program will be the breaking down of traditional barriers between specialized engineering fields, the development of a mathematics program geared to the opportunities of the age of computers, and introduction of the professional problems of engineering at an early stage in the student's education.

The new Carnegie Grant follows a $150,000 grant, for a five-year period beginning in 1952, from the Carnegie Corp., New York, which led to a study of and development in the humanities program at Case. This program was also supported by grants from the Fund for the Advancement of Teaching of the Ford Foundation.

SCIENCE FELLOWSHIPS MULTIPLY

The **National Science Foundation** announced award of 525 fellowships, in addition to 922 awarded last month, in the natural sciences and allied fields. The additional awards were made possible when the President signed the Foundation's supplementary appropriation bill recently approved by the Congress.

Fellowship awards are made in furtherance of the Foundation's policy of encouraging gifted college graduates, established scientists, and college teachers of science to obtain advanced training in the sciences on a full time basis.

FOUNDRY EDUCATION PROGRAM

Wheelabrator Corp. of Mishawaka, Ind., through the Wheelabrator Foundation, recently established a $100,000 foundry educational program with the Foundry Education Foundation in commemoration of Wheelabrators 50th anniversary of service to the foundry industry.

The program is established to assist young men in obtaining an education at the graduate level in the fields of engineering, science, marketing, and business management. Through this plan, a core of high-level specialists will be developed for leadership in the foundry industry. This program augments Wheelabrator Foundation's undergraduate scholarships at Purdue and Notre Dame and also suppliments it's contributions to the Associated Colleges of Indiana.

Fifty graduate fellowships will be available—one for each of the fifty years of Wheelabrator Corp.'s association with the foundry industry. Each graduate fellowship will be in the Amount of $1500. The total value of these fellowships will be $75,000.

The grant also puts $25,000 at the disposal of the FEF for fellowships, scholarships, or other educational purposes that may be recommended by FEF Trustees for approval by the Wheelabrator Foundation.

CERAMIC RESEARCH

Expressing great satisfaction with results already accomplished at the **School of Ceramics, Rutgers University**, on the use of lead compounds in ceramic bodies under a grant made by the Lead Industries Association last year, Robert L. Ziegfeld, Association Secretary, has announced a new grant to Rutgers which will more than double the funds available for the work this year. It is under the direction of Dr. John H. Koenig, director of the School of Ceramics.

This is part of the aggressive research program on the use of lead in all phases of ceramics which the Association is sponsoring as a result of the great interest the ceramic industries are showing in lead compounds. At least nine graduate research fellowships will be supported this year in ceramic schools from coast to coast.

Among the studies will be included work on low-firing porcelain enamels for steel and aluminum, phase diagrams involving lead oxide, glazes for whitewear and clay sewer pipe, glazes for brick, properties imparted to glass by lead, and lead ferrolectrics.

HIGH TEMPERATURE PROJECT

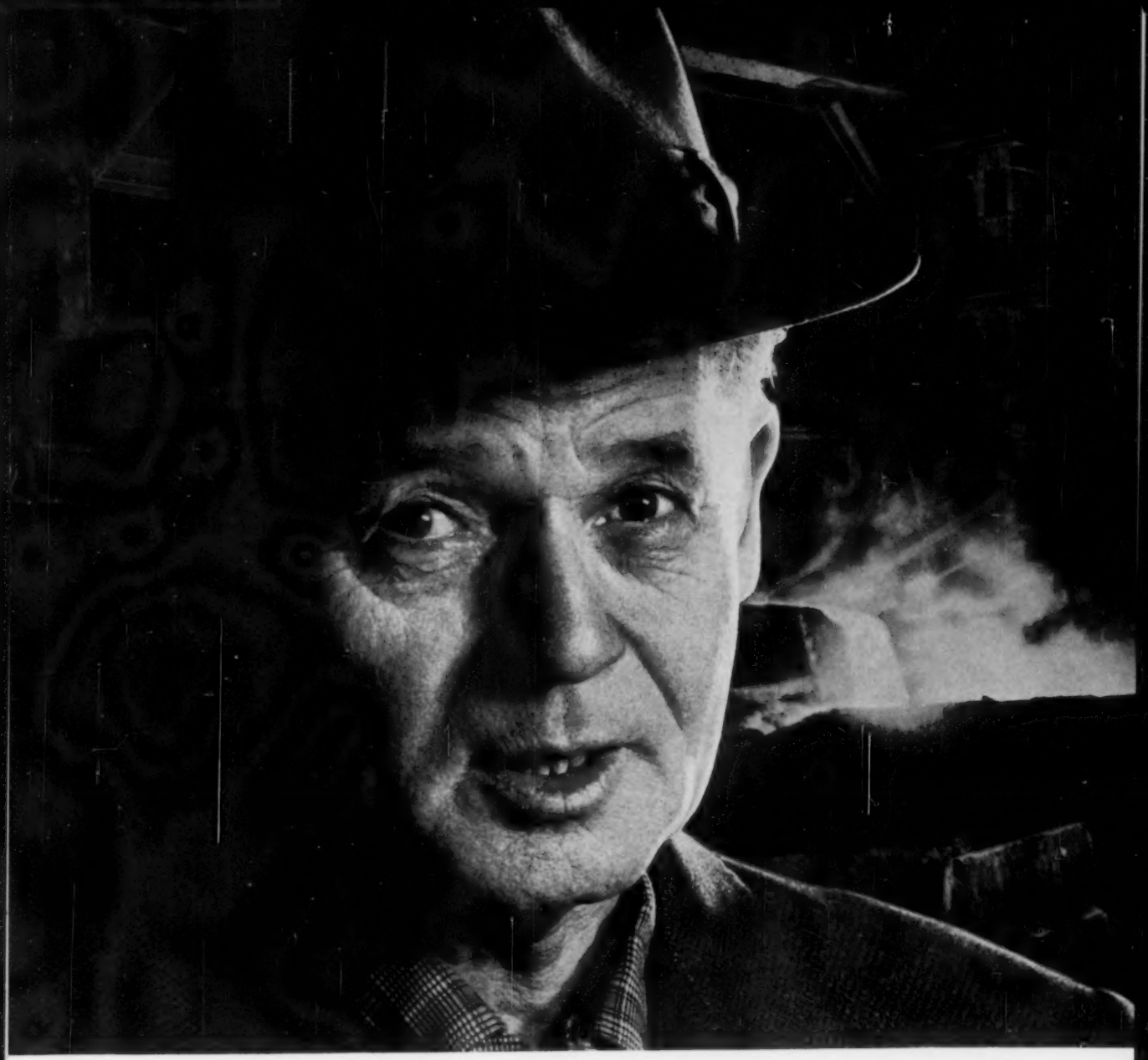
The National Science Foundation has awarded a 2½-year grant for research on the *Imperfections of Metals at Elevated Temperatures*, to the department of metallurgy at the **Northwestern University Technological Institute.**

Directing the project under the $25,400 NSF grant will be assoc. prof. John O. Brittain. He said the research project will be conducted in a new mechanical metallurgy laboratory being developed as part of the Technological Institute's expansion of research facilities.

The new laboratory is set on a foundation separate from the remainder of the building, and has its own insulation, making it particularly adaptable for conducting highly sensitive creep experiments at both very high and very low temperatures.

Purpose of the project will be to determine by experiment, the effect of imperfections on the creep of metals. Research will involve subjecting metals to stresses at temperatures ranging from 1100° to

(Continued on page 431)



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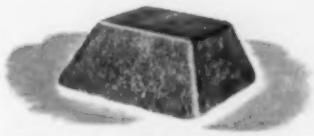

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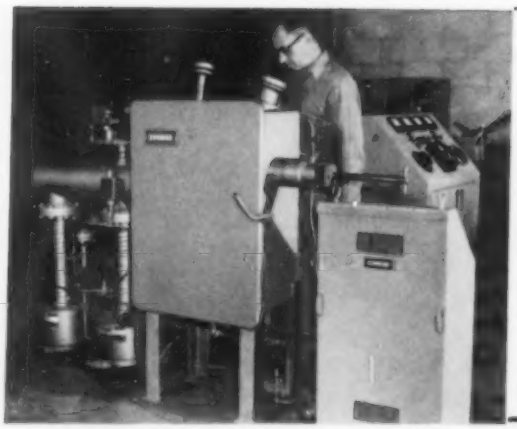

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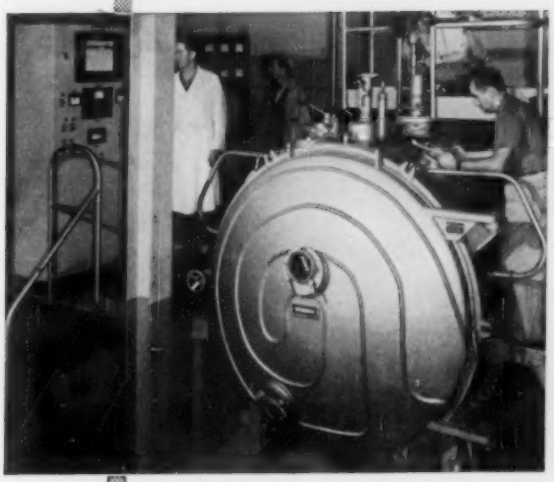
Stokes offers laboratory facilities for vacuum melting and heat-treating. This installation is available as a tool to those interested in learning more about vacuum melting and heat-treating of ferrous or non-ferrous metals, as well as those looking for opportunities to make special vacuum investigations. Set-ups can be made to simulate production cycles, and serve to develop comprehensive recommendations for future production and systems operation procedures.

Opportunities

Vacuum Furnaces



The General Thomas J. Rodman Laboratory of the Watertown Arsenal, Watertown, Mass., uses this 500 lb. capacity Stokes Vacuum Furnace for research on Titanium, other reactive metals, and radioactive metals. Special locks enable additions to be made or samples taken without breaking the vacuum.

treating, sintering, brazing and stream degassing. In addition, special systems can be custom-designed to meet the exact needs of specific operational requirements. All-the-way service, before and after the sale, completes the picture of how well Stokes is equipped to work with you.

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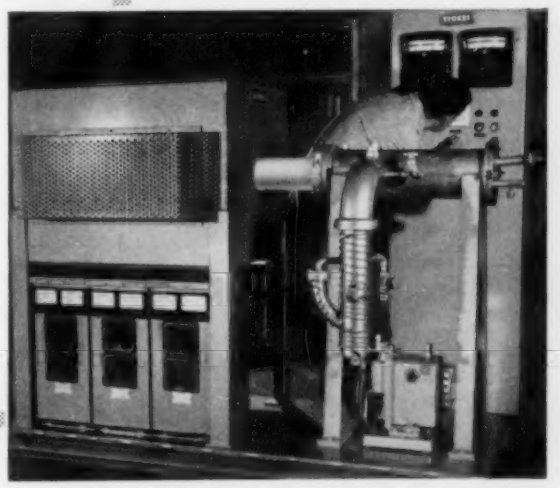

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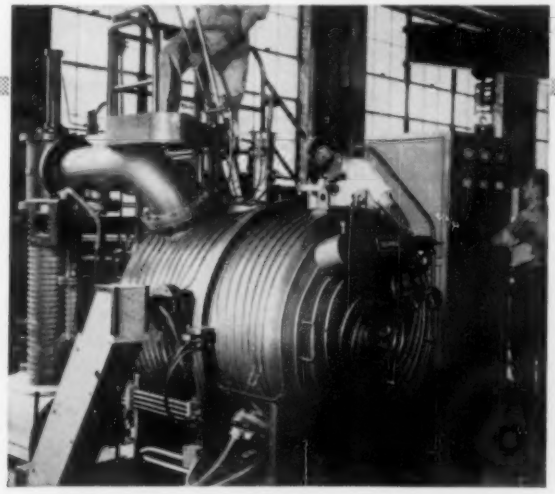

Stokes Vacuum Furnace, in the Bayonne, N.J. laboratory of International Nickel Co., Inc., is used to study the effects of vacuum melting and casting on nickel base alloys destined for jet engines and other extreme-temperature services. It has a 50-pound crucible.

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Lower prices for hyper-pure silicon

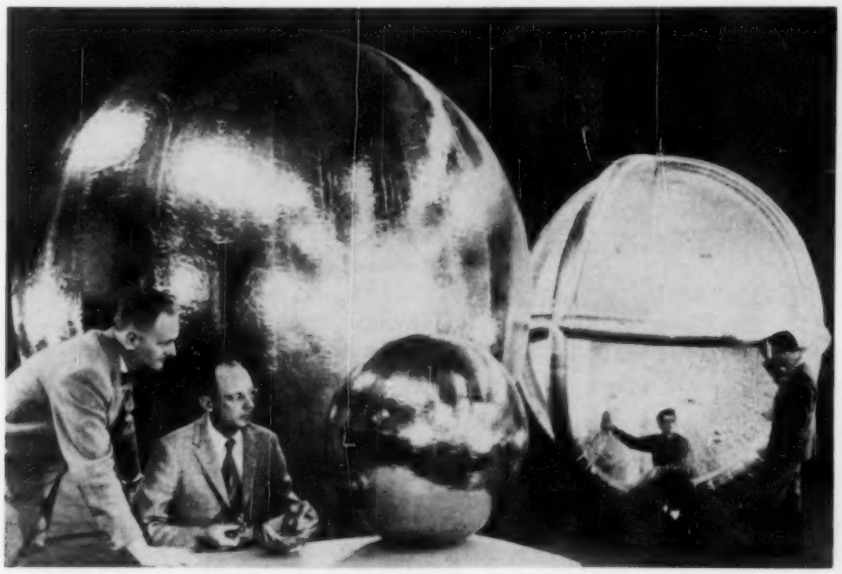
Lower prices for all grades of hyper-pure silicon were announced recently by du Pont, coincident with the start-up of the Nation's first full-scale silicon plant near Brevard, N. C. Semi-conductor grades range from $155 to $320 per lb, while solar-cell grade is selling for $90. Solar grade silicon was developed by du Pont for use in solar batteries which convert the sun's rays into electrical energy. In the Vanguard satellite, for example, silicon helps to provide power for transmitting scientific data back to Earth.

Molybdenum metal plant

Construction of a $1 million plant for the production of molybdenum and molybdenum-base alloys was announced by American Metal Climax, Inc. Situated at Coldwater, Mich., the new plant will include two vacuum-arc consumable electrode melting furnaces, developed especially for molybdenum production. The plant will also include machining, forging, and auxiliary facilities for the production of billets and bars. Initial production will be largely material to be used for guided missile components.

Foil satellites to be launched

One of the most visible and least expensive US satellites for future launching is a 12-ft sphere made with 0.005-in. thick Reynolds aluminum foil, which is sandwiched to a 0.00025-in. thick sheet of plastic film. For launching, such spheres will be folded into a small package together with a regular satellite. When the latter is hurled into orbit, the foil sphere would be detached and inflated by a bottle of nitrogen. After the sphere has rounded itself out, the nitrogen would be allowed to escape, and the gleaming spaceball would orbit. Because it is so big and yet so light, the slightest bit of air resistance would greatly affect it, making it possible to compute air drag and density. There is discussion about putting 100-ft diam models into orbit, for with four or five such spheres orbiting, radio signals could be bounded around the earth.

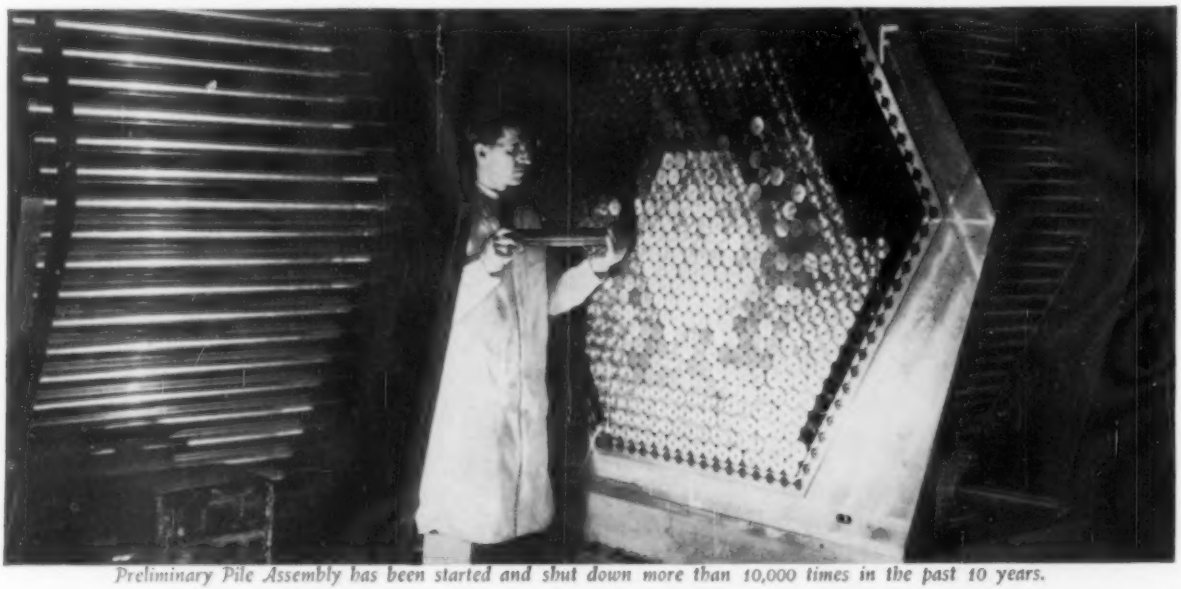


Pictured are three of the aluminum foil sub-satellites which are scheduled to be launched into outer space. Large sphere in left rear is a 12-ft model, while on the table rests a 30-in. diam version. Sphere's designer, William J. O'Sullivan of the National Advisory Committee for Aeronautics (seated) holds collapsed 30-in. sphere. Right background is 12-ft diam corner reflector for reflecting radar waves.

furthering creative technology at

The KNOLLS ATOMIC POWER LABORATORY

To men concerned with pushing back the frontiers of reactor technology, the many facilities for fundamental and applied research at KAPL hold a special interest. Among the unique critical assemblies for experimental purposes is the PPA, *pictured below*. This Preliminary Pile Assembly can be shut down and restarted in 15 minutes in order to rearrange fuel elements to simulate different reactor designs. Only last year two new buildings were completed to house two additional experimental reactors, a 704 computer, and to provide modern office facilities for an enlarged mathematical staff.



Preliminary Pile Assembly has been started and shut down more than 10,000 times in the past 10 years.

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EDITORIAL

SECRETARY KIRKENDALL SPEAKS ON TRANSACTIONS

by Ernest Kirkendall, AIME Secretary

JUST over a year ago, the three Branches of the Institute became Societies, and each was charged with providing those services that it felt its members wanted most, within the Bylaws of the Institute and a balanced budget for each Society. Another step forward in providing those services effectively yet economically took place this year.

Transactions papers have been completely separated from the JOURNAL OF METALS. They are available to members who are interested at the nominal price of $5.00 per year, in the form of a new bimonthly journal, TRANSACTIONS OF THE METALLURGICAL SOCIETY OF AIME. The price to the public is $20.00 per year.

By separating these papers, they need now only be sent to those members who desire them. This eliminates a considerable waste circulation among members who are concerned primarily with non-technical operations. Unlike the former Supplement to the JOURNAL OF METALS, transactions papers from the Extractive Metallurgy and Iron and Steel Divisions are now published in the TRANSACTIONS, so that it is truly the *technical* journal of the Society. As former-President Kinnear said in his foreword to the first issue, " . . . since the science of metals is basic to all, it is hoped that ideas originating from one Division will provide inspiration and answers to problems of members of the other Divisions."

This move by The Metallurgical Society is similar to one made by the Society of Petroleum Engineers of AIME several years ago, whereby their volume giving statistics of oil and gas production, which was a service of considerable value to a certain portion of their membership, was made available for sale to members rather than supplied to all without additional charge.

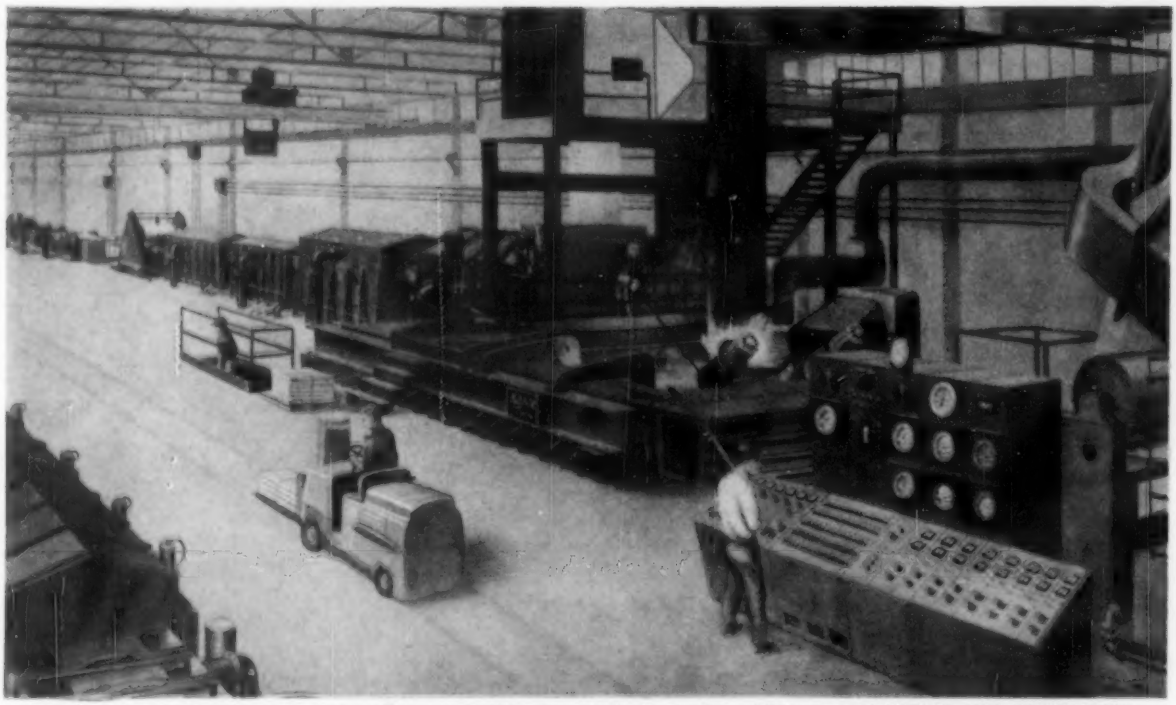
For many years, the mining Divisions of the Institute helped to pay for the publication of these metallurgical papers. The support that they have given has always been deeply appreciated. In recent years their own publication programs left them without sufficient funds to continue this practice. During recent years, industry has been subsidizing the Metals Research Publication Fund to the extent of about $35,000 per year.

It is now imperative that The Metallurgical Society finance their own publications program. Until a self-supporting program can be established, industry, through the Metals Research Publication Fund, will continue to subsidize the program. But this is only a temporary situation, as it is not felt that industry can be prevailed upon to continue this support year after year. Their cooperation is a high compliment to the quality of the papers appearing in the TRANSACTIONS. Although any individual paper may be of direct use only to a limited number, its eventual importance to the metallurgical profession may be tremendous.

Starting in January of this year, members of The Metallurgical Society were advised of the availability of the new bimonthly TRANSACTIONS OF THE METALLURGICAL SOCIETY, for $5.00 per year. As these announcements are often overlooked in the rush of everyday work, it is hoped that this will serve as clarification to members who may have been confused. Members of Mining and Petroleum Societies may of course also obtain subscriptions at the member rate.

The February and April issues have already been mailed out, but additional copies were printed so that late subscribers may obtain complete sets of this new publication.

Research is the life-blood of our profession, and it is the professional duty of all metallurgists to keep abreast of new technical developments in their fields as reported in the TRANSACTIONS.



The above is a partial view of the two continuous galvanizing lines at the Martins Ferry, Ohio, plant of WHEELING STEEL CORPORATION. Both lines use AJAX 60 cycle induction galvanizing furnaces and zinc premelt furnaces. The main galvanizing furnace shown holds 175 tons of zinc, is rated 2000 kw, and produces over 40 tons per hour at speeds in excess of 300 feet per minute. These continuous galvanizing lines produce WHEELING's patented SOFTITE sheet.

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AEC-tr-2875 *Table of Coulomb Phases and Amplitudes Taking into Account the Finite Nuclear Size*, by L. A. Sliv and B. A. Volchok, Academy of Sciences, USSR, 25¢, 21 pp., 1956.

AEC-tr-2872 *Second All-Union Conference on Photosynthesis*, Div. of Biological Sciences, Academy of Sciences, USSR, and Faculty of Soil Biology, M. V. Lomonosov State University, 60¢, 122 pp., 1957.

AEC-tr-2925 *All-Union Conference on the Application of Radioactive and Stable Isotopes in the National Economy and Science* and *Abstracts of Papers, First All-Union Conference on Radiation Chemistry*, Academy of Sciences, USSR, $2.20, 451 pp. (1 vol.), 1957.

PB 121782 *Research on Effects of Prestraining and Notch Sharpness on the Notch Strength of Materials*, by G. W. Geil and N. L. Carwile, National Bureau of Standards for Wright Air Development Center, U. S. Air Force, $3.00, 113 pp., 1956.

PB 121512 *Thermal Buckling*, by F. V. Pohle and I. Berman, Polytechnic Inst. of Brooklyn for Wright Air Development Center, U. S. Air Force, $1.00, 37 pp., 1956.

PB 111963 *Corrosion Properties of Various Materials in High Temperature Waters*, by C. J. Lancaster and W. L. Williams, U. S. Naval Engineering Experiment Station, 75¢, 22 pp., 1953.

PB 121098 *Applications, Properties, and Fabrication of Thermenol Type Alloys*, by J. F. Nachman and W. J. Buehler, Naval Ordnance Laboratory, $1.50, 52 pp., 1956.

PB 121837 *Investigation of Mechanical Properties, Corrosion Resistance, and Oxidation Resistance of Thermenol, an Iron-Aluminum-Molybdenum Alloy*, by K. L. Kojola, U. S. Naval Gun Factory, 75¢, 29 pp., 1955.

PB 121901 *Attaching Thermocouples by Capacitance Welding*, U. S. Naval Gun Factory, 50¢, 8 pp., 1955.

PB 131007 *Symposium on Preservation for Mobilization Requirements*, U. S. Naval Civil Engineering Research and Evaluation Laboratory, $8.00, 524 pp., 1956.

PB 121026 *Development of Lean-Alloy Chromium-Nickel Stainless Steels for High Temperature Use*, Cornell Aeronautical Laboratory Inc. for Bureau of Aeronautics, U. S. Navy, $2.25, 86 pp., 1954.

PB 131009 *The Effect of Various Heat Treatment Cycles Upon the Mechanical Properties of Titanium Alloys With Various Interstitial Levels*, by B. F. Hadley, G. W. Bauer, and D. Evers, Mallory-Sharon Titanium Corp. for Wright Air Development Center, U. S. Air Force, $5.50, 220 pp., 1957.

PB 121885 *Wear Studies With Titanium*, by R. J. Benzing and A. N. Damask, Wright Air Development Center, U. S. Air Force, 50¢, 20 pp., 1957.

PB 121972 *The Effect of Microstructural Variables and Interstitial Elements on the Fatigue Behavior of Titanium and Commercial Alloys*, by C. B. Dittmar, G. W. Bauer, and D. Evers, Mallory-Sharon Titanium Corp. for Wright Air Development Center, U. S. Air Force, $2.50, 96 pp., 1957.

PB 121978 *Relaxation Behavior of Titanium Alloys*, by F. J. Gillig, Cornell Aeronautical Laboratory Inc. for Wright Air Development Center, U. S. Air Force, $2.25, 87 pp., 1956.

PB 121833 *True Stress-True Strain Properties of Titanium and Titanium Alloys and Effect of Vacuum Annealing on the Impact Properties of Titanium and Titanium Alloys*, Watertown Arsenal Laboratories for U. S. Army, $2.25, 90 pp., 1955.

PB 121819 *Tantalum Determination*, by R. W. Moshier and J. E. Schwarberg, Wright Air Development Center, U. S. Air Force, $1.00, 39 pp., 1956.

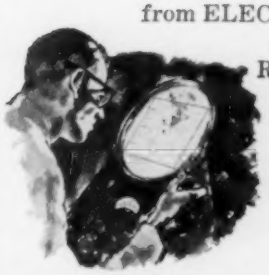
PB 121928 *Study of Ultra High Temperatures*, by C. S. Stokes, Research Inst. of Temple University for Air Force Office of Scientific Research, $1.00, 31 pp., 1956.

PB 121930 *Study of the Utilization of Solar Furnace for High Temperature Research on Solids*, by T. E. Tietz and N. K. Hiester, Stanford Research Inst. for Air Force Office of Scientific Research, 50¢, 10 pp., 1956.

PB 121889 *Temper Brittleness of Boron-Treated Steel*, by S. J. Rosenberg, National Bureau of Standards for Wright Air Development Center, U. S. Air Force, $1.75, 66 pp., 1956.

PB 121950 *Development of Cast Iron-Base Alloys of Austenitic Type for High Heat-Resistance and Scale-Resistance*, by F. Eberle, J. H. Hoke, and W. E. Leyda, Babcock & Wilcox Co. Research Center for Wright Air Development Center, U. S. Air Force, $2.50, 99 pp., 1957.

from ELECTROMET Metals Research



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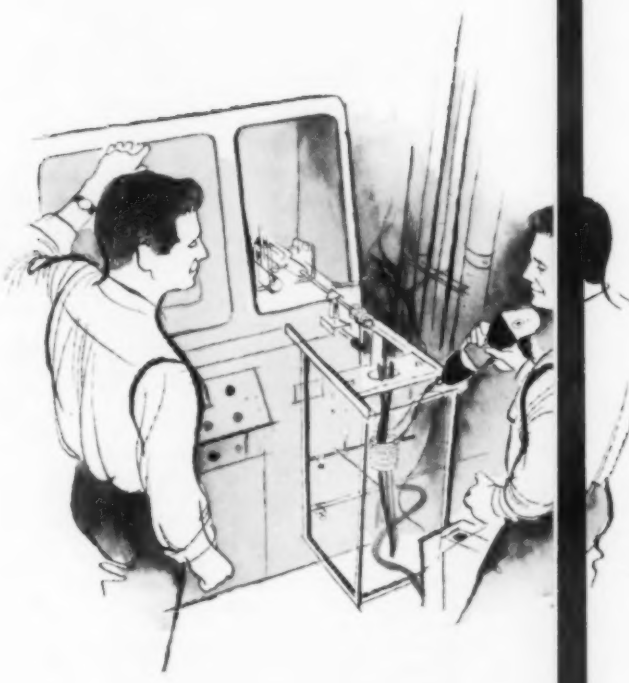

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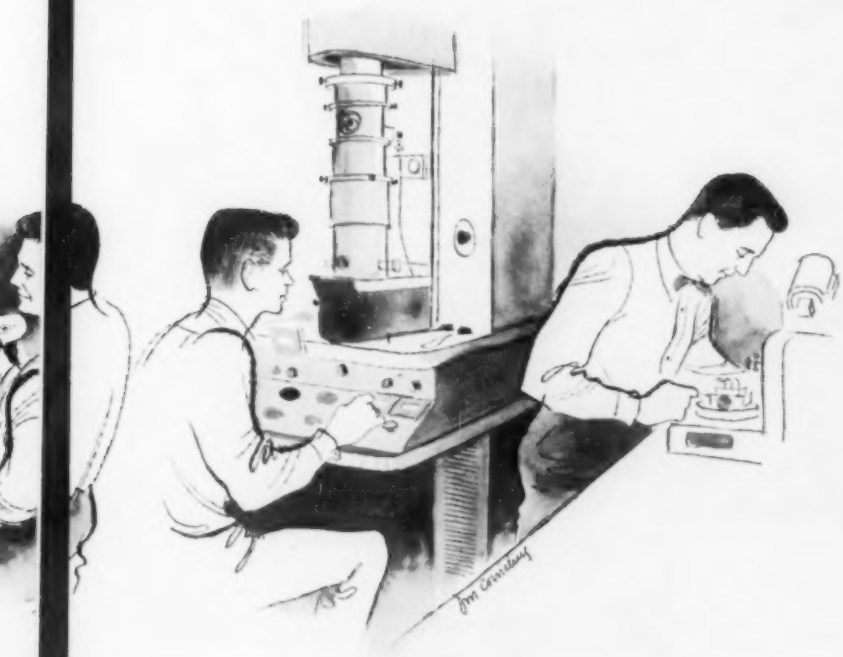

In control material development, research is under way on compounds and alloys with boron, silver, and indium.

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FLASH SMELTING COPPER CONCENTRATES

Flash smelting is a comparatively new method of smelting copper and nickel sulfide concentrates. It takes place when the concentrate, with or without additional fuel, is suspended in gases containing oxygen whereby the heat of oxidation reactions bring the suspended particles to a smelting temperature. From the suspended state the hot particles are separated and molten slag and matte are deposited on the furnace hearth.

by Petri Bryk, John Ryselin, Jorma Honkasalo and Rolf Malmstrom

THE theoretical possibilities for the realization of flash smelting have been known for a long time. Calculations concerning the same can be found in previously published literature[1], and suggestions for its accomplishment in practice are described in patents on the subject. Likewise a smaller number of practical test results are known.

Autogenous smelting is closely connected with flash smelting, since in flash smelting techniques it is possible to use the heat evolved by oxidation of the sulfides for the smelting process to such an extent that under certain conditions smelting can proceed without additional fuel. However, in the event that insufficient heat is available from the oxidation of the sulfide concentrates, then additional heat may be supplied by preheating the air, using oxygen enriched air, on burning additional fuel with the concentrate.

Techniques for flash smelting

a) Reverberatory and Cubic Furnaces

Klepinger, Krejci and **Kuzell,**[2] have suggested that fine grade ore be smelted in an apparatus resembling the reverberatory furnace so that air and concentrate suspension is blown horizontally from the front end of the furnace through the burner, in which case the temperature required for smelting is obtained by the oxidation of concentrate and by the burning of additional fuel, if required. Air is supplied in "proper proportion (preferably heated)". It is furthermore stated that "over oxidation may be corrected by introducing unoxidized material (coarse if desired) into the furnace, on the hearth of which the smelted material collects". This method resembles that of the ordinary pulverized-coal fired reverberatory smelting method, except that concentrate instead of pulverized coal is fed into the burner. See Fig. 1a.

PETRI BRYK, JOHN RYSELIN, JORMA HONKASALO, and ROLF MALMSTROM, are associated with Outokumpu Oy, Finland. Paper presented at AIME Annual Meeting, New York, February 1958.

Krejci holds a patent[3] for a similar process, with the exception that he provides for wide, air nozzles below the burners. The purpose is to supply a flow of high velocity air which carries the suspension of concentrate, thus preventing the particles from reaching the bath too soon. In this way the particles are carried further in the combustion zone and have more time for reaction. Note Fig. 1b.

The method patented by **Freeman**[4] differs greatly from the foregoing—Fig. 1c. The patent claims show that the length and width of the smelting furnace equal its height. The concentrate burner is in the center of the roof and directs the suspension of concentrate, fuel, and flux with air downward through the furnace roof. The gaseous products of combustion are removed through the furnace in the direction countercurrent to the movement of ore and oxidizing gas. It is suggested that the heat content of the waste gases be used for preheating the air.

None of the above three methods have, as far as it is known, been realized on a commercial scale. In the Klepinger et al. system it is evidently difficult to obtain enough time for the concentrate particles to remain in air suspension so that the desired oxidation of sulfides, with a simultaneously good utilization of oxygen, can be achieved. As already mentioned, a large amount of excess air is detrimental to the heat economy of the flash smelting process. It is also obvious that the fine fraction of the concentrate follows the gases, thus resulting in heavy dust losses. In the system suggested by Krejci, the reaction time for the suspended particles will be longer, but the high air velocity will surely increase the amount of excess air as well as the dust losses. Freeman's suspension flow, directed downward against the molten surface, is advantageous for the separation of the particles but, on the other hand, the leading of exhaust gases against this flow is bound to result in a considerable mixing of the gases. In this way, a part of the suspension flow will be carried with the exhaust gases out of the furnace.

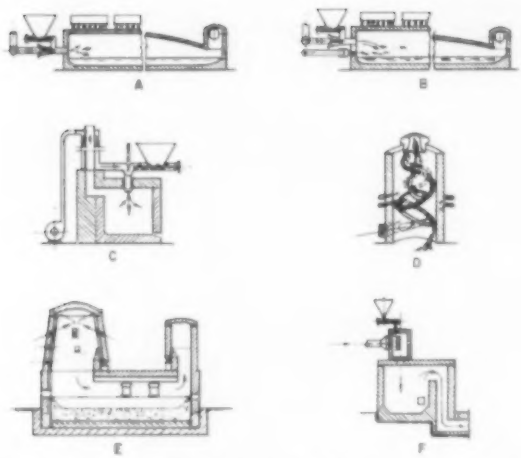


Fig. 1—Equipment for flash smelting: A, by Klepinger; B, by Krejci; C, by Freeman; D, the Saint-Jacques furnace; E, Pilot furnace used by Leiserovitch; and F, Pilot furnace used by Tonkonogi.

b) Roasting type furnace

The Saint-Jacques Furnace is known primarily as a sintering and roasting apparatus. Fig. 1d shows how the furnace operates for roasting purposes. By means of the distributor on the top of the cylindrical furnace, the air and concentrate suspension is brought into downward rotational motion. At a lower level there are tangential nozzles for secondary air and additional fuel. The gases leave the furnace through the center of the distributor. The smelting of copper concentrate with the Saint-Jacques furnace was tried at the Bor smelter in Yugoslavia before World War II, but oil consumption was larger than the concentrate fed. The results were so negative that further experiments reportedly were not performed.

c) Reaction shaft

As reported by **Leiserovitch** et al,[5] interesting flash smelting pilot plant tests were made in the USSR in 1935. The pilot plant furnace comprised a reaction shaft and a settler part, Fig. 1e. In the upper part of the slightly conical reaction shaft there were four concentrate burners at the same level through which the concentrate and air suspension is blown into the furnace. They were inclined in such a manner that a rotating motion of the gas resulted. Close to the concentrate burners, four oil burners were arranged in a similar manner. When leaving the shaft, the suspension flow was directed downward against the bath in the settler, and the gases left the furnace through the uptake. Three different test runs, the longest of which lasted 185 hr, are reported. During these test runs 14 to 18 tons of concentrate per day, with a moisture content of 3 to 4 pct, were smelted in the furnace. Oil consumption was about 100 to 140 kg per ton of charge smelted. The report shows that there was very heavy wear on the magnesite bricks, especially in the connection between the reaction shaft and the settler. They also had trouble with drying the concentrate and with keeping the feed rate constant. In various parts of the flue after the uptake, molten flue dust had a strong tendency to clog up the flue, similar to difficulties which seem to have appeared wherever flash smelting tests have been made. No reports concerning further testing in the field of flash smelting with this type of furnace in the USSR, are known.

Lange of Bergakademie Freiberg, Eastern Germany[6] has made laboratory tests on flash smelting poor, complex, copper-zinc-lead sulfide concentrates in a furnace with about 3 ft high reaction shaft. A reducing atmosphere was maintained in the furnace, resulting in volatilzation of more than 90 pct of both lead and zinc, while in the settler the rich matte and the slag were easily separated. Gas burners on the top of the reaction shaft were used for heating the furnace, as well as for obtaining the reducing conditions needed. Judging from these laboratory-scale results it seems to be possible to achieve a good separation of zinc and lead from copper by the flash smelting of bulk concentrates under reducing conditions.

d) Cyclone Chamber

As reported by **Tonkonogi** et al.[7] tests have been made in the USSR with complex concentrates, similar to those used by Lange, and which cannot be selectively floated. Smelting is carried out in a cyclone smelting chamber, similar to the combustion chambers used by coal-fired cyclone furnace boilers, as can be seen from Fig. 1f. The air is blown tangentially with a velocity of 200 to 460 ft per sec into the cyclone chamber about 17 in. in diam. and 31 in. high, inside dimensions. The concentrate is fed into the cyclone through the roof in such a manner that, at the level of about one third of the cyclone height, it is thrown against the walls, i.e. into the high-temperature reaction zone. From the reported 20 tests, each of which lasted 6 to 13 hr, it can be seen that the smelting process is possible in such a cyclone, and that without reducing conditions, about 50 pct of the lead and zinc are removed, the rest remaining in slag and matte. There is very heavy wear of the lining in the cyclone chamber, and the cyclone can be kept in operation only by means of water cooling and natural lining. This means that heat losses in the cyclone smelting

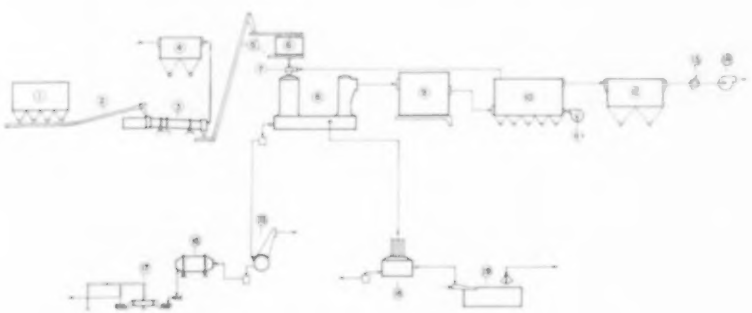


Fig. 2—Flowsheet of the Harjavalta smelter: 1, concentrate storage; 2, belt conveyor; 3, dryer; 4, dryer cottrell; 5, Redlers; 6, feed hopper; 7, concentrate burner; 8, flash smelting furnace; 9, waste heat radiation boiler; 10, heat exchanger; 11, primary air fan; 12, cottrell; 13, damper for automatic draft control; 14, gas fans; 15, converter; 16, anode furnace; 17, casting wheel; 18, slag cleaning furnace; and 19, slag granulating.

chamber will be high. Even if the dimensions of the cyclone chamber are small, its total heat losses will be so high that it seems to be difficult to achieve an autogenous smelting process. On the other hand, the additional heat needed can be recovered in the form of steam from the cooling system of the cyclone smelting chamber. In the above article as well as in that by Shulzinger,[9] it is mentioned that in the near future pilot plant tests with this cyclone smelting method will be made at some Soviet smelter.

Commercial-scale flash smelting

The first flash smelting method which has been realized on a commercial scale is the **Outokumpu process.**[9] It has been operated on a commercial scale at Harjavalta, Finland, since 1949. The Ashio Smelter of the Furukawa Mining Co., which went into operation in 1956, has also adopted the Outokumpu system.

In this process oxidation reactions are carried out in a vertical shaft, in a downwardly flowing concurrent system. Preheated air and dried concentrate in proper proportions are fed into a burner on the top of the reaction shaft. The mixing of air and concentrate takes place effectively in the burner, and the resulting suspension is directed vertically downwards into the shaft in such a manner that it is spread over the whole shaft area. When the particles enter the hot shaft under these conditions, ignition takes place instantaneously. By exothermic reactions the temperature of the particles is raised to the smelting temperature. At the same time, the partial pressure of oxygen decreases, so that by leaving the reaction shaft it will be close to zero. In this way, a good utilization of oxygen is achieved, provided that the dimensions of the reaction shaft are correct.

Separation of the particles from the gas takes place when the gases are turned through an angle of 90°, upon leaving the reaction shaft. Gases containing mostly sulfur dioxide and nitrogen are exhausted from the furnace at a temperature slightly above the slag temperature.

The particles are collected in the molten bath in the settler, which is located horizontally below the reaction shaft. Iron oxide and silica from the feed react, forming slag and at the same time, molten matte drops are separated and collected on the furnace hearth. The matte grade is easily controlled by the ratio of air to concentrate. By altering the air preheat temperature, the temperature in the furnace can be kept on a desired level. If the concentrate to be smelted has a composition which requires air preheat temperatures too high to be reached practically, the remaining heat can be provided by burning extraneous fuel, together with the concentrate, in the reaction shaft.

Another flash smelting system which is now in operation on a commercial scale, is the International Nickel Co.'s process.[10] Instead of preheated air, the INCO process uses almost pure oxygen to eliminate the heat deficit in the system. The combustion of sulfide concentrate is carried out in a similar manner, as suggested by Klepinger, i.e. in a horizontal flow. Obviously, the rich oxygen atmosphere accomplishes such a reaction velocity that the desired oxidation of sulfide particles can be obtained before the particles reach the molten bath. This INCO process has been in operation at Copper Cliff since 1952.

Concentrate handling at Harjavalta

At the Harjavalta smelter, concentrates and fluxes from storage bins are fed into an oil-fired, refractory-lined kiln for drying. The kiln, which was manufactured by F. L. Smidth & Co., Copenhagen, has a length of 66 ft and an inside diam of 6 ft; the first section is equipped with lifter blades. The capacity of the kiln is 660 tpd when drying from 7 to 0 pct moisture. The oil consumption under summer conditions is 1.7 US gallons per short ton of dried concentrate, which corresponds to 1800 BTU per lb of evaporated water.

The concentrate is transported from the dryer at 230°F to the furnace by means of three different Redler chain conveyors, as can be seen from the flow sheet, Fig. 2. All of these Redlers are manufactured by Gebrüder Bühler, Uzwil, Switzerland; they have proven very successful for the transportation of this hot and very fluid material. The third Redler, which transports the concentrate to the furnace, is used for controlling the feed rate. Since continuous feed control is a very important factor in the flash smelting system, this type of a volumetric feed control with a Redler has, after various trials, been considered to be the most practical one. The chain speed is variable within the limits of 3 to 20 ft per min.

The flash smelting furnace

There are two identical flash smelting furnaces at Harvajalta. At the start it was not known what the refractory failures in the reaction shaft would be; therefore, two furnaces were built. Later, however, it became obvious that only one furnace was necessary, and the other furnace has been in reserve for years.

The flash smelting furnace consists in principle of a reaction shaft, a settler, and an uptake, Fig. 3. The reaction shaft is a 26 ft high by 12 ft inside diam cylinder, lined with chrome-magnesite brick. In the lower part of the shaft are two water-cooled rings connected to the flanges, which support the lining

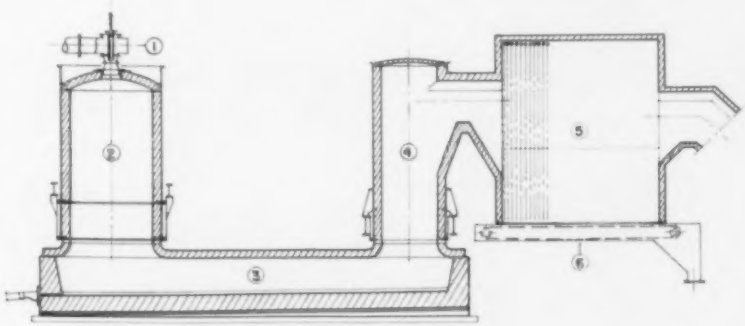


Fig. 3—Harjavalta flash smelting furnace and waste heat boiler: 1, concentrate burner; 2, reaction shaft; 3, settler; 4, uptake; 5, radiation waste heat boiler; and 6, traveling grate.

of the shaft. Originally the diameter of the lower part of the shaft was reduced, thus giving it a conical shape, as described by Benitez.[11] The conical part of the reaction shaft was intended to improve the separation of molten particles by the increased velocity. This, however, resulted in heavier brick wear in the conical part and consequently, in higher heat losses through the necessary water cooling. After studying this problem by scale model tests, the reaction shaft was rebuilt cylindrically without a constricted discharge. After this alteration of the shaft, the wear on the bricks has decreased; dust losses are about the same as before the alteration.

The settler has the following dimensions: 56 ft length, 13 ft width and 5.6 ft height. The roof (sprung arch), side walls, and bottom are magnesite lined. It is essential that the furnace bottom, which is made as an inverted arch, has small heat loss; therefore, it consists of the following brick layers: magnesite, fireclay, and insulation. Under these layers refractory concrete connects the settler with the steel construction, which, in turn, is laterally supported by buckstays.

As far as the brick wear of the settler is concerned, the roof has not been renewed since the furnace was put into operation in 1949. The long life of the arch, compared with standard reverberatory practice, can be explained by the fact that in the flash smelting process the temperature of the gases is about 2370°F, whereas the flame temperature in the reverberatory furnace is correspondingly much higher. In the side walls, especially in the front end, there is corrosion on bricks at the slag line, and this must be repaired locally about once a year. The best brick at the slag line and in the lower part of the reaction shaft has proven to be burnt chrome magnesite brick. The bottom of the furnace has an indefinitely long life.

From the settler the gases exit through the uptake, the inside diameter of which is 8 ft, and the height from the arch to the center of the discharge opening is 23 ft. This uptake must be cleaned at intervals of a few months.

Waste heat boiler

From the uptake, the gases at 2325°F are led to a forced-circulation-type, wall-radiation, water boiler. Each furnace has its own boiler, the main function of which is to cool the gases containing molten dust particles to such a temperature (1565°F) that the particles solidify and do not sinter and cause clogging in the duct. The inside dimensions of this boiler are: total height 26 ft, width 14 ft, and length 23 ft. The walls are lined with 1¼ in. outside diam steam tubes (1 11/32 in. from center to center). A part of the tubing in the second half of the boiler consists of superheater tubes. In this boiler the steam pressure is 570 psig and the superheating temperature 680°F, the generated steam being used in a turbogenerator set. This radiation boiler has operated very satisfactorily. The flue dust which settles in the boiler does not clog and can be easily removed by blowing with compressed air once or twice per shift.

Heat exchanger

The hot gases at 1565°F are led from the boiler through a 6½ ft inside diam, brick-lined duct to the heat exchanger. The purpose of the heat exchanger is to preheat the air needed for smelting to a sufficiently high temperature, by using the available heat content of the waste gases. The heat exchanger, which is a metallic recuperator, operates according to the counter current principle; the gases flow outside, and the air inside of heat exchanging tubes. As can be seen from Fig. 4, the heat exchanger is built in two parallel sections both of which comprise 11 groups. Each group consists of 56 heat-resisting, cast-steel tubes. Due to the dust content, the outside of the elements is without fins. The inside of the elements is finned, except in the cold end of the heat exchanger where surface temperature of the elements, if finned, would fall below the dew point, causing corrosion. In order to remove the dust from the tube surface, automatic swinging chain cleaners are used. In addition, air lancing is employed in the hot end once or twice a shift.

Cottrell and exhaust gas fumes

From the heat exchanger the gases are led to a hot cottrell. It is essential that when leaving the heat exchanger the temperature of the gases be high enough to stay above the dew point, so that corrosion in the cottrell can be avoided.

The present cottrell was built in 1953 and consists of two parallel precipitator chambers. Two mechanical high-voltage rectifiers supply the electric current, one for the first half and the other for the second half of each of the chambers. An automatic mechanical rapping system, which operates without cutting out the high voltage, efficiently cleans the collecting and emitting electrodes. From the cottrell, the gases are led to the exhaust gas fan center in which three parallel-connected fans take care of the transportation of the gases from the flash smelting furnace to the sulfuric acid plant. In connection with these fans, there is an automatic draft control which operates dampers so that about *zero inch* water gauge pressure prevails in the flash smelting furnace.

Matte and slag treatment

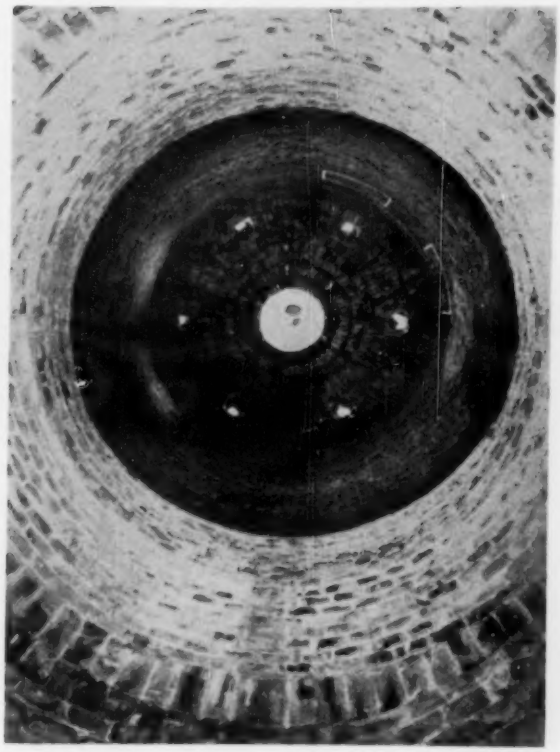
The matte grade is usually above 60 pct. It is transferred to the converter department which consists of three 11½ ft by 21½ ft converters. Only one is used at a time for blowing the flash smelting furnace matte.

The resulting blister copper is transferred into two oil-fired, rotating anode furnaces. Anodes are cast in copper molds on a 14-mold rotating casting wheel, and then cooled in water.

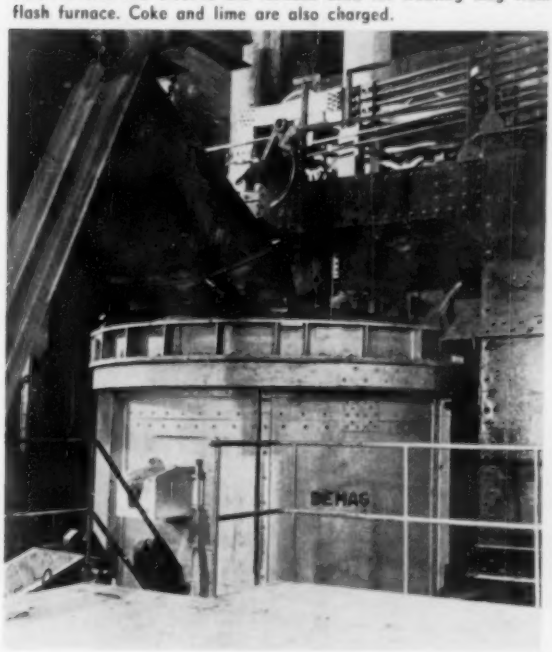
When flash smelting is used, it is generally necessary to produce a high-grade matte to fully utilize the oxidation heat of iron and sulfur; therefore, the copper content of the slag is comparatively high.

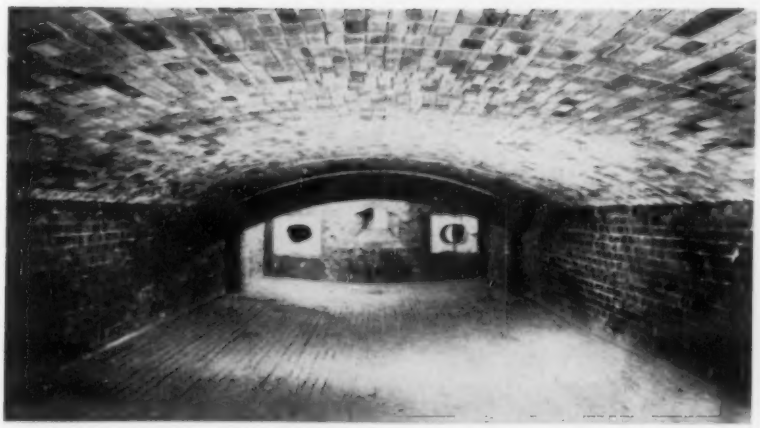
In order to obtain the highest possible copper recovery, and also to recover nickel and cobalt from the slag, a special slag cleaning process has been developed. The principle of this new process is based on the fact that the iron:oxygen ratio in fayalite-type slag has a decisive influence on the remaining metal content of the slag. It has been proven that by altering this iron:oxygen ratio, e.g. by a reducing treatment of the slag, it is possible to remove copper, cobalt, and nickel from the slag. This reducing treatment is preferably carried out in an electric furnace, because a desired degree of reduction can easily be obtained and controlled. With the addition of lime and coke, calcium carbide is formed in the electrode zone under proper electrical conditions. The calcium carbide reacts in-

Right—Interior of reaction shaft of Harjavalta flash furnace lined with chrome-magnesite brick. Center hole is for concentrate burner, while openings around perimeter are for oil burners.

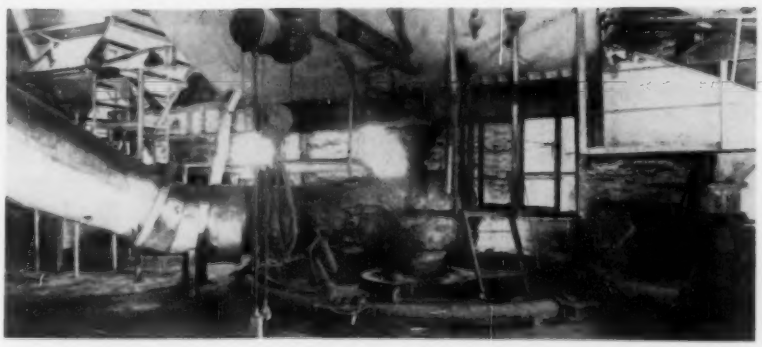


Below—4000 kva electric arc furnace used for treating slag from flash furnace. Coke and lime are also charged.





Left—At Harjavalta, the interior of the settler of the flash furnace is similar to a reverberatory furnace. Newer design furnaces have full suspended arch roofs.



Right—Concentrate burner at top of reaction shaft of flash furnace.

Table I. Operating Statistics for the Harjavalta Smelter, October 1957

Typical Analyses, Pct	Cu	Fe	S	SiO_2	Ni	Co	Zn	CaO
Concentrates	20.7	31.8	33.2	4.8	0.18	0.13	3.22	
Flux				80				
Matte	61.6	12.0	22.3		0.33	0.12	1.5	
Slag (Dump)	0.53	38.0	1.2	29.7	0.09	0.21	4.7	1.2

Flash Smelting Furnace Operation

Concentrate smelted, metric tons	14,492
Flue dust (obtained and re-fed), metric tons	878
Oil in flash smelting furnace kg per metric ton concentrate	14.2
Oil in concentrate dryer kg per metric ton concentrate	6.3
Steam generated kg per metric ton concentrate	380
Air preheating temperature, °C	510

stantaneously with the slag and effects the desired reducing action. Instead of lime, crushed iron-silicate slag or iron oxide and silica can be used whereby the intermediary reducing agents are ferro silicon and silicon carbide, which have the same effect as calcium carbide. The process is carried out to the point where the oxygen ratio in the fayalite slag corresponds to the equilibrium point in the system fayalite-metallic iron-oxygen. It is not necessary to proceed beyond this point where actual formation of metallic iron starts. With this treatment, copper, nickel, and cobalt are efficiently removed from the slag and collected on the furnace bottom. It is advantageous to add a small percentage of sulfide in order to convert the bottom layer to a matte with a low melting point.

Tests with an 800 kw electric furnace have been made for some time. These tests show that the copper content of the flash smelting slag easily can be reduced to 0.2 pct. Simultaneously, the cobalt and nickel contents reach a level under 0.1 pct, which means that an unusually high nickel and cobalt recovery can be obtained. The power consumption in these tests has been about 130 kw-hr per ton of slag treated. The consumption of coke has been between 1 and 2 pct of the amount of slag treated.

On the basis of these positive results, it was decided to use this process for treating all the smelter slag. As the existing electric settling furnace was not suitable for this special slag cleaning process, a new electric furnace was erected and put in use in December, 1957. So far, the slag cleaning results have been up to expectations.

The Ashio flash smelting plant

The Furukawa Mining Co. in Japan decided in 1954 to modernize their copper smelter in Ashio. After a thorough study of the Outokumpu flash smelting process in Harjavalta, they considered it to be the best solution in their case and resolved to adopt the system.

The new smelter which started up in March 1956, operates on the same general principles as the Harjavalta smelter, although the choice of the auxiliary equipment is different in some cases. The operations at Ashio smelter are described by Okazoe of the Furukawa Co. in an article recently published.[12]

Summary

The standard pyrometallurgical process for copper usually comprises three stages: roasting, smelting, and converting, which are generally carried out in different metallurgical units. Roasting and converting are both markedly exothermic processes, whereas smelting is the heat consuming part of the process. In standard practice a large amount of the heat of the exothermic processes is lost, whereas flash smelting is characterized by the idea that two of the three processes are carried out simultaneously in one unit, the flash smelting furnace. This leads to a considerable saving of fuel and, depending on the composition of the concentrates, the smelting can even be carried out autogenously.

In reverberatory smelting the heat content of the waste gases is normally recovered, but the sulfur eliminated in the furnace normally cannot be recovered economically because of the excessive dilution of the gas. Contrary to this, the flash smelting furnace gases are, after passing the waste heat boiler, an ideal feed for the sulfuric acid plant because of the steady flow and unusually high sulfur dioxide content. As there are practically no sulfur losses, except in the slag, sulfur recovery is very high.

Experience has also shown that the refractory lining in the flash furnace, especially the roof, has a much longer life than in a reverberatory furnace. A considerable saving is also effected in the converter operation, because flash smelting normally produces a high-grade matte whereby the converters require less compressed air, and the slag blowing stage is shortened. For this reason the converter lining can also operate under less strenuous conditions.

Arthur G. McKee & Co. is licensed by Outokumpu to engineer smelters using their process in the Western Hemisphere.

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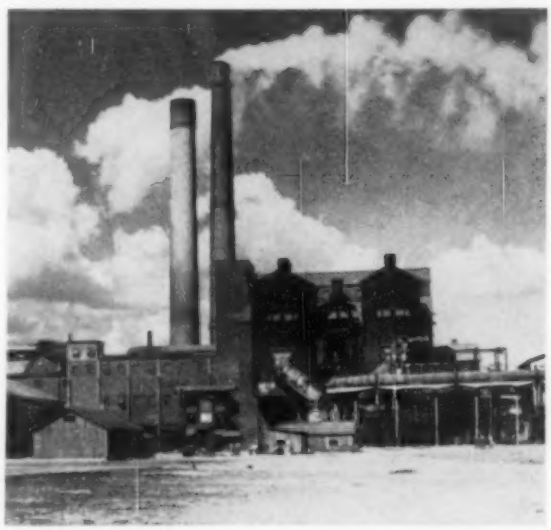
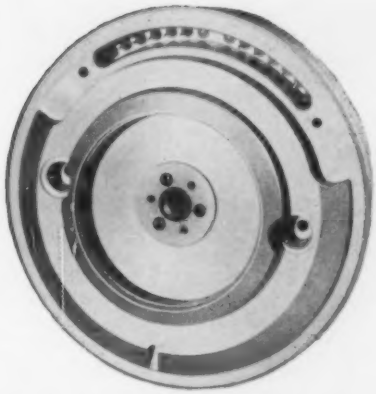


The Harjavalta Smelter at Outokumpu Oy, Finland. In 1944 the smelter was moved from eastern Finland and erected in 7 months. Center buildings house waste heat boilers which were built in 1949.

BERYLLIUM:



PRESENT AND POTENTIAL USES

Beryllium is not new; it has been available as a metal in small quantities for the past 25 years. However, until the AEC developed new uses for beryllium in atomic reactors, very little attention was paid to the metal.

by Lawrence F. Boland

BERYLLIUM is one of the metals which owes its recent prominence and expansion of production to the growth of atomic energy. Prior to this, the industry grew almost entirely around beryllium-copper alloys.

In 1951, the AEC established facilities to produce beryllium metal in volume—both as vacuum-cast billet and hot-pressed shapes. This was a big step for beryllium, boosting production from several hundred pounds a year to approximately 40,000 lb annually. Late in 1956, the AEC contracted with two private producers for delivery of a total of 200,000 lb a year for five years, beginning in 1958. The new plants constructed to meet this requirement are the only large-scale, basic beryllium metal plants in production in the US today, for the AEC-owned facility for production of beryllium billet has been shut down and placed on a standby basis.

Beryllium metal

Practically all of the beryllium metal in nuclear reactors is used as a reflector or moderator, in order to capitalize on the low thermal neutron absorption cross-section of beryllium. For example, the new AEC Engineering Test Reactor at Idaho Falls, Idaho, has, surrounding the core of the reactor, an inner reflector of beryllium. Its primary purpose is to reflect neutrons from the chain reaction back into the core, thus reducing the amount of fuel needed and increasing reactor efficiency. In Europe, further use has been made of beryllium in nuclear reactors as an additive to magnesium alloys for fuel element cladding.

In addition to its uses in nuclear reactors and in alloying copper, the metal beryllium is now being considered by private industry for many varied applications. These include: parts for X-ray cameras and high-speed cameras, aircraft brake parts, underwater instruments, accelerator targets, as a *getter* material for use in heated liquid metals, gas turbine engine parts, high-frequency electronic parts, aircraft instruments, additives to aluminum, zinc and other alloy metals to act as a deoxidizer, high-velocity testing equipment, additives to steel, rotor shafts in refrigeration equipment, and rocket engine components.

The Air Force and the aircraft industry have also indicated substantial interest in beryllium for structural applications in aircraft and missiles, because beryllium is the only light metal with a high melting point and extremely high modulus of elasticity. This is shown in Table I, which compares the physical properties of beryllium with other aircraft metals. The property which beryllium lacks is ductility, which naturally limits use of beryllium in large quantities.

Currently planned and under way are many research and development programs on beryllium, representing several million dollars of sponsorship by the Air Force. Programs contracted for to date include evaluation of additives to beryllium, brittle-phase investigation, and sheet rolling. Planned for release shortly are further research contracts on casting, composite materials, joining, and surface effects, as well as programs on development of forging and extrusion methods.

Beryllium oxide

Beryllium oxide is a *sleeper* in the sense that few people are aware of present uses, entirely separate from the uses for the metal. But indications are that beryllium oxide might rapidly parallel beryllium as a construction material for use in atomic reactors.

Beryllium oxide is a refractory material with rather unique properties. Current prime interest, from an

LAWRENCE F. BOLAND is Vice President, The Beryllium Corp., Reading, Pa. This paper was presented at the Atomic Energy Management Conference in Chicago, March 17, 1958, and will appear in the conference proceedings.

application standpoint, is in its use as a moderator and/or reflector, chiefly to take advantage of its high melting point (twice that of beryllium metal). Hot-pressed and machined shapes of beryllium oxide are now being produced commercially to densities as high as 2.85 g per cu cm (approximately 95 pct of theoretical) for use in nuclear reactor construction. Comparison between some of the properties of beryllium oxide and beryllium is provided in Table II.

Unlike most ceramic materials, which are poor heat conductors, beryllium oxide possesses high thermal conductivity. It also possesses good insulating properties for high-frequency electrical currents. These characteristics suggest many applications in electrical and electronic fields, such as Klystron and ceramic tubes, for which low dielectric losses and high thermal shock properties are required.

Work is being done on the use of beryllium oxide for power tube components to increase power output of the tubes and reduce the size of cooling units. Beryllium oxide is also being used as a slurry for the coating of graphite crucibles where carbon contamination is undesirable. It is being studied for use in antenna window applications, airborne radar antennae, electronic parts, high-temperature thermocouples, protection tubes cooling power transistors in electronic equipment, and guided missile components, such as gas seals.

Furthermore, beryllium oxide is radiation stable and does have high-temperature characteristics, including good thermal shock properties. This should suggest many applications to the electronic-nucleonic engineer.

After examining the established and potential uses for beryllium metal and beryllium oxide, the obvious question is,

Why isn't beryllium now being used in more tonnage by the aircraft industry, in electronics, and other fields?

The answer lies in four limiting factors: cost, availability, brittleness, and toxicity.

Cost

Although demand has brought the cost of beryllium metal down from the $100-$150 per lb for which it sold in the late forties, it is still an expensive material. Today's prices are around $47 per lb for billets and $65 to $80 per lb for hot-pressed and rough machined blocks. It has been estimated that, if present annual volume holds and an additional 200,000 pounds of beryllium a year were required, the price could be reduced another 15 pct to 20 pct.

A much larger reduction in the price of beryllium to nuclear users can be made by improvements in fabrication. Today, practically all beryllium parts for nuclear reactors are produced by powder metallurgy. This means that vacuum-cast billet must be lathe chipped, and subsequently ground into powder. The blended powder is hot pressed under vacuum into blocks. These blocks, in turn, are then machined to the particular shapes required. This procedure is costly.

Now that the Air Force has expressed increased interest in beryllium for non-nuclear uses, we in the industry are studying methods to produce beryllium mill forms, i.e. sheet, rod, bar, extrusions, etc., by conventional methods. We hope to be able to develop techniques whereby the lowest cost beryllium *bead* can be converted into a billet which can be directly extruded or hot formed. Preliminary work done to date is very encouraging.

Currently, practically all of the beryllium oxide parts are being produced by hot pressing techniques. However, we are also exploring cold compacting, slip casting, extrusion, and other techniques that may permit more flexibility in design and reduce costs of end pieces. Although beryllium oxide shapes of simple design now cost $40 to $60 per lb, better manufacturing methods may make available shapes for moderator/reflector end use in another five years at $15 to $20 per lb.

Availability

When beryllium is reviewed for possible use in new fields, one of the negative features mentioned is its scarcity. The beryllium industry has never been put to a test, but it is confident that substantial increases in the supply of beryllium can be found as new uses develop. Beryllium constitutes 0.0005 pct to 0.0006 pct of the earth's crust, which compares with boron's 0.0017 pct.

Beryllium is found in over 30 minerals, but at present, beryl ore is the only mineral used as the source of beryllium. Future beryl-supply prospects appear favorable, according to a 1954 report from the US Bureau of Mines. Estimated reserves in Brazil, the largest beryl-producing country, have increased recently, and estimates of the beryl reserves of the tin-spodumene belt in North Carolina have been increased to 823,000 tons, nearly three times the amount formerly estimated.

With regard to production capacity for beryllium metal, two new plants, designed for a minimum of 100,000 lb a year each, have sufficient capacity to supply all near-future beryllium requirements. Needed, however, are more stimulating programs on beryl ore procurement and more effort to make low-grade domestic beryl ore useable to industry.

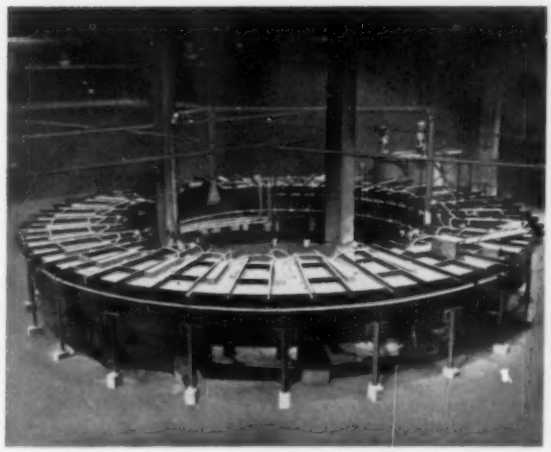
Ductility

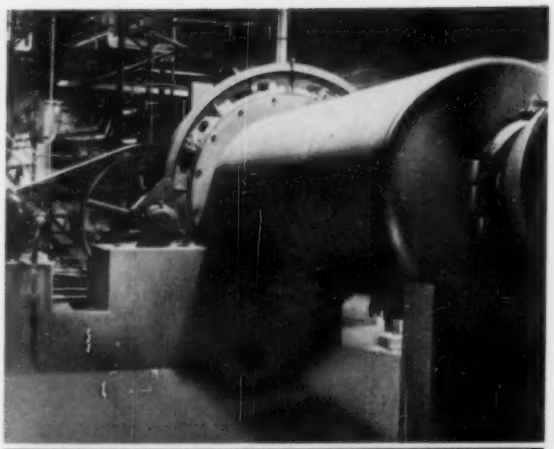
A third problem which has deterred beryllium metal from wider usage is that current grades of beryllium lack *three-dimensional ductility*. This prevents the large-scale fabrication of beryllium by conventional metal-working techniques, and the solution of this problem is the primary objective of most of the research work being conducted on beryllium today.

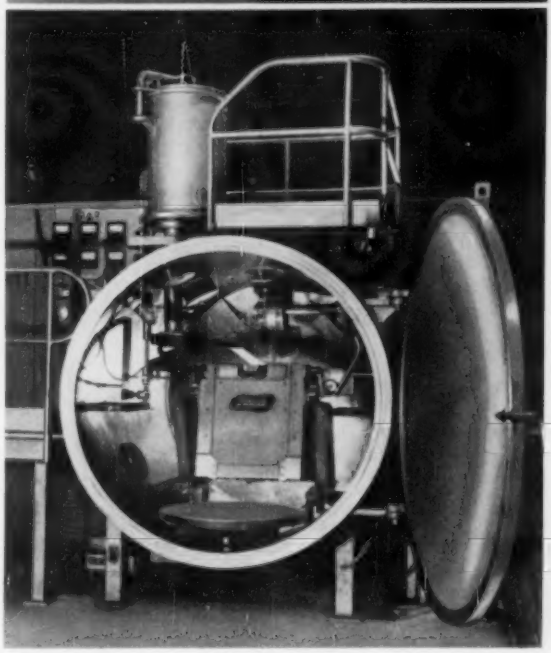
But the problem of brittleness is one of degree only, and it does not prevent the use of present-grade beryllium in aircraft and missiles. If further research and development yields methods for producing beryllium metal with greater three-dimensional ductility, then substantially larger fields of application must inevitably result.

Toxicity

The health hazard associated with improper handling has constituted another great barrier to the wider use of beryllium metal and beryllium oxide. There is no question about the possible toxic problem, but it is not insurmountable. Beryllium can be handled safely with proper controls. Today many Government installations—and more recently an increasing number of industrial concerns—have seen fit to learn more about beryllium, and they are now fabricating beryllium parts in their plants.







ABOVE: Beryl ore sintering furnace used in the initial stages of production of beryllium metal.

UPPER RIGHT: A rotary pebble mill wet crusher for reducing the size of the reacted beryl ore and chemicals.

RIGHT: Induction-heated vacuum melting and refining furnace used in beryllium production. Crude beryllium shot is charged into a 120 lb capacity crucible; melting and refining is done at a pressure of about 10μ. Five ingot molds can be accommodated on the turntable at the bottom of the furnace for pouring under vacuum. Product is better than 99 pct Be.

More fabricators needed to supply growing market

There is today a need for additional fabricators of beryllium metal components to supply both nuclear fields and non-nuclear markets—not only aircraft prime and subcontractors, but companies whose normal business is forging, precision machining, stamping, manufacturers of fasteners, instrument manufacturers, and many others. General interest in beryllium as a new material is advancing rapidly, and current research will create future demand for sources of beryllium components. If beryllium is to be a widely used metal, the industries mentioned and others must become familiar with its properties and learn how to work with it.

New markets must be developed before reduced costs of beryllium to the nuclear user can be expected. However, engineers who are considering the use of beryllium and beryllium oxide in nuclear power reactors over the next five to ten years, have reason to look forward to better grades of beryllium, better manufacturing techniques, and lower costs.

Table I. Physical Properties of Beryllium Compared with other Aircraft Metals

	Density, Lb per Cu In.	Melting Point, °F	Modulus of Elasticity, 1,000,000 Psi	Tensile Strength* 1000 Psi
Beryllium	0.066	2350	40.0	40 to 80
Aluminum	0.100	1220	9.6	25 to 85
Magnesium	0.065	1202	6.5	24 to 52
Steel	0.280	2750	30.0	185 to 250
Titanium	0.165	3300	15.5	130 to 170

* Estimates based on sheet material comparisons.

Table II. Properties of Beryllium Oxide and Beryllium Metal

	Melting Point, °F	Thermal Neutron Absorption Cross-Section, Barns	Hardness	CuCm Density, G per
Beryllium	2345	0.009	Rb 55 to 90	1.85
Beryllium Oxide	4658	0.010	2000 (Knoop)	3.025

For the first time

TOUGH-PITCH COPPER CONTINUOUSLY CAST

Described by some as a breakthrough and by others as the first major development in copper refining in 50 years, Asarco's new roasting-melting-continuous-casting unit is, without a doubt, a big step forward in copper production.

by F. Weston Starratt

CONTINUOUS casting of tough-pitch copper, long a goal of the nonferrous metals' industry, is now an everyday fact at the Perth Amboy, N. J., plant of American Smelting & Refining Co. The integrated unit for roasting and melting copper cathodes and casting them into cakes up to 25 ft long is capable of handling the refinery's entire output of 14,500 tons per month. Wire bars are also being cast, and later this year it is anticipated that billets too will come from the new unit.

This is, of course, not the first time that copper has been continuously cast. Asarco, and other firms, have, for almost 30 years, been casting oxygen-free grades of copper and copper alloys in high heat-conductivity graphite molds. But over 90 pct of the copper consumed in the U. S. is tough-pitch copper, containing 0.03 to 0.04 pct O_2. At the copper casting temperatures, the graphite mold would react with the oxygen contained in the tough-pitch copper, thereby rendering its use impossible.

Longer copper cakes required

For many years Asarco research people have been attempting to surmount the impasse, since there has been a mounting demand from copper fabricators for long-length cakes which could most efficiently be used in high speed sheet and strip mills.

Standard size cakes, up to the present time, have been produced in sizes 25 to 36 in. wide, 4.5 to 7 in. thick, and 36 to 92 in. long. But these have been too short to serve the new rolling mills with maximum efficiency. Furthermore, vertically cast copper cakes longer than 37 in. have shown poor surface quality. Thus, the key to the problem appeared to be a technique for continuous casting cakes to any length desired.

Mold design, the crux

At the crux of the matter of continuous casting of tough-pitch copper cakes was the design of a high heat conductivity mold, which would not be affected by the oxygen content of the copper at casting temperatures. Success in perfecting such a design, led first to the construction of a pilot plant and later to the installation of a commercial-scale unit.

This new technique, while specifically designed to cast tough-pitch copper, is so versatile that it can be used, with but minor modifications, to produce cakes of oxygen-free, phosphorized, or other special coppers. While in actual fact the operation is semi-continuous, it does permit the casting of copper cakes in lengths up to 25 ft, weighing up to 17,000 lb. The thickness is generally 5 in., and widths of 25, 30, or 36 in. can be produced in the two-mold unit which is now in operation. Cast lengths can later be cut to any length desired by the customer to feed his particular sheet or strip rolling mill.

While the actual design of the mold is a secret, pending final approval of Asarco's patent application, it is known to be a compressed graphite mold, with thin walls, cooled by water on the exterior. The reason for the lack of reaction between the graphite and the oxygen in the copper lies in the high heat conductivity design of the mold which keeps the temperature of the mold sufficiently low. How this is achieved, Asarco will not say, but it is felt by this author that the mold probably contains high-conductivity metallic elements, possibly water cooled.

The mold is 15 in. deep. It has been designed so that it has a heat dissipation rate of about 1500 hp. The design enables the mold to solidify molten copper, cast at 2075°F, at a speed of 20 in. per min for a 5x36 in. cake, or the equivalent of 1150 lb of copper per min.

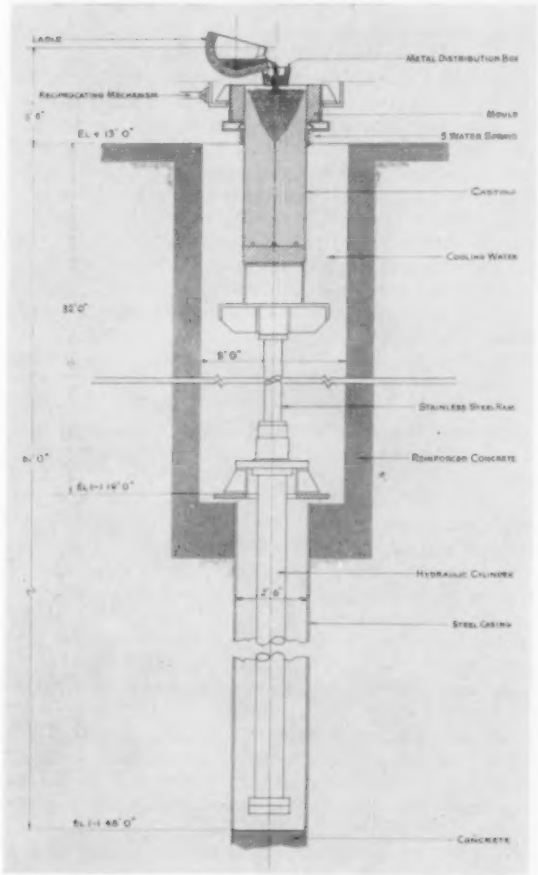
To prevent the cast copper from sticking and to insure a smooth cake surface, the entire mold assembly is oscillated by a reciprocating gear. The vertical stroke is a small fraction of an inch, and the variable speed driving motor can operate up to several hundred cycles per minute.

Casting into 61 ft well

The casting pit is a water-filled well, 61 ft deep. At the top of the well are two of the bottomless graphite molds. At the beginning of the pouring operation, a stainless-steel ram with a 27 ft stroke is moved up into the mold and sealed to its sides by asbestos filling material. After the first 8 in. of copper have solidified, the oscillating mechanism is started, and the cake is lowered into the water-filled well at the rate of 20 in. per min. As the solid cake emerges from the bottom of the mold, it is cooled by a series of high-speed water-spray jets. The pit in which the hydraulic piston operates is kept full of water which is pumped into each mold at the rate of 1600 gal per min. The heat dissipation capacity of the entire cooling system is rated at 5800 hp.

After the cake is cast, the mold is swung to one side, and the cake is pushed about 3 ft out of the pit. Then a 10-ton crane with specially designed

Asarco's unit for the continuous casting of copper cakes up to 25 ft in length and weighing up to 17,000 lb.



hoisting scissors lifts the 17,000-lb cake to a nearby platform. There it is inspected and cut to specified lengths by a saw. The first 6 in. of each cake has poor surface quality, due to splattering in the mold; it is discarded and remelted.

The continuous cake casting unit has a capacity of some 160 tons of copper per 8-hr shift.

Billet and wire-bar production

The melting furnace installed at the new Perth Amboy casting plant has been designed so as to provide a continuous flow of molten copper to the cake-casting molds. However, because of fluctuations in orders, it has been necessary to provide an alternate casting unit into which the molten metal may flow. Thus, at the present time, there is a casting wheel of the standard design for casting wire bars.

Furthermore, copper billets for making tubes will soon be produced by the new plant, since Asarco is going ahead with construction work on two lines for the continuous casting of copper billets. Details have not yet been released, but the operation is expected to start up sometime during the summer.

New-type melting operation required

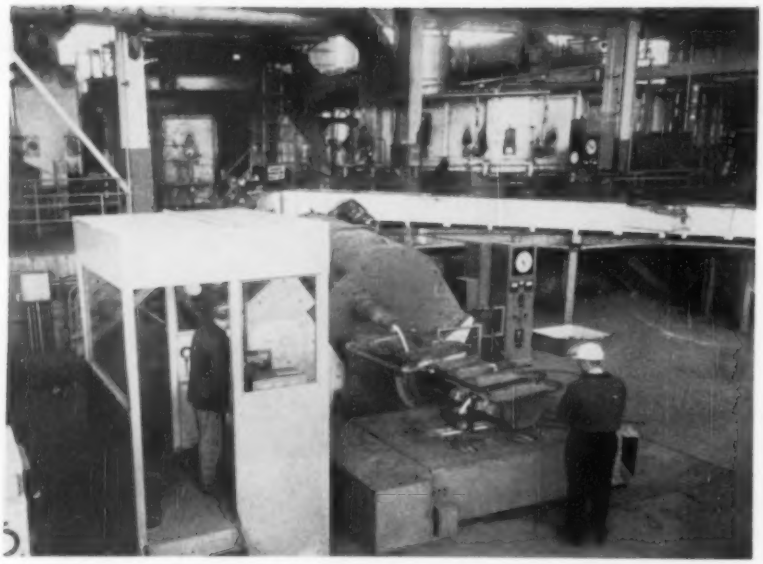
Not only is the high heat-conductivity mold an essential part of this continuous casting operation, but equally necessary is a continuous flow of molten copper of low sulfur content.

The existing process for melting down cathodes in a reverberatory furnace is a batch process, requiring the casting of the entire contents of the furnace, amounting to as much as 400 tons, in no more than 5 hr. The reverberatory furnace functions chiefly to oxidize the sulfur, picked up by the copper from the sulfate electrolyte in the refining tanks. A final poling operation is essential to reduce the oxygen content of the copper to proper levels. Once this has been attained, casting must proceed as rapidly as possible to prevent too great a pick-up of sulfur from the furnace fuel. The traditional casting wheel is capable of casting shapes at a sufficiently rapid rate to accommodate the reverberatory furnace. But such a batch operation is not satisfactory for continuous casting.

Engineers at Asarco selected the arc furnace as being capable of providing a continuous flow of molten copper. But this type furnace could not be used by itself for melting copper cathodes for two reasons: 1) high electric power costs (about 1 cent per kw-hr) at plant location, and 2) the arc furnace did not promote the removal of sulfur. It was decided, therefore, to use the arc furnace in combination with another furnace designed to eliminate sulfur and use low-cost fuel.

Pre-heating furnace

The oil-fired pre-heating or roasting furnace used at Perth Amboy was designed by the Salem-Brosius firm. It is a rotary furnace into which piles of copper cathodes are charged on pallets. It requires approximately 1 hr for these pallets to make a complete revolution in the furnace at which time the sulfur is largely volatilized, and the cathodes have attained a temperature of 1500°F. The cathodes are removed from the pre-heating furnace and immediately charged into the adjacent electric-arc furnace by the same charging machine. Savings in energy costs represented by this pre-heated charge are significant.



Shown at left is the world's first plant for continuous casting of tough-pitch copper cakes. At upper right is the oil-fired pre-heating furnace for volatilizing sulfates on cathodes and pre-heating them for charging into the arc furnace, shown at the upper left. In the foreground is the tundish and casting mechanism, which stands directly over a 61 ft pit into which the cast billet descends.

Largest direct-arc copper furnace

Providing the molten copper for the new plant at the rate 30 tons per hr is an 18-ft diam Whiting electric-arc furnace with a bath capacity of 90 tons. Its transformer is rated at 8000 kva with a maximum rated current of 30,400 amps. The furnace employs three 18-in. diam graphite electrodes, each of which, when assembled, weighs 4000 lb.

One problem encountered in the furnace design was that of accurate control of the arc. The arc discharges from the bottom of the electrodes to the molten bath and requires the maintenance of an air gap of 0.5 to 1 in. at all times. Ripples, due to charging or surges in the bath, require an extremely sensitive control system to permit the electrodes to adjust automatically to changing conditions at the metal surface. If the electrodes are lowered too far, a short circuit results, and if they are withdrawn excessively, the arc is snuffed.

Obtaining the necessary sensitivity of control in combination with rapid motion of the massive electrodes and their holders was a formidable requirement. It was accomplished by utilizing a rotary regulator and controls actuating an hydraulic servo system, which was designed to Asarco's specifications by the Whiting Corp. and General Electric Co.

Except for additions of electrodes, and infrequent periods of slagging, door cleaning, or adjustment, the furnace can be operated continuously. The molten copper flows continuously from the furnace at about 2125°F by displacement and by adjustment of the furnace tilt angle.

Oxygen control

From the arc furnace, the molten copper flows down a covered launderer which splits into streams destined for cake or wire-bar casting. Each stream passes through a 10-ton Ajax electric induction holding furnace, which serves as a surge tank as well as for regulating temperature and oxygen content.

The arc furnace is capable of producing molten copper of negligible oxygen content, due to the general reducing atmosphere of the furnace; however, the furnace is equipped with a draft system, permitting a controlled quantity of air to sweep through the furnace. At Perth Amboy this is operated so as to permit the copper to pick up from 0.01 to 0.02 pct O_2 by the time it flows out of the furnace. During its passage through the launderers and holding furnace more oxygen is picked up, so that by casting time the oxygen content is at the desired 0.03 to 0.04 pct for tough-pitch copper.

Furnace economics compared

The entire installation at Perth Amboy, including the pre-heating furnace, the arc furnace, and the casting units, is designed to produce on a continuous basis, low-sulfur, tough-pitch copper in the shapes and sizes which are most economic for fabricators.

While the economics of such a continuous operation may appear obvious, several comments may amplify this point. For equal yearly production, the capital cost of a large arc furnace and pre-heating furnace is less than that of multiple reverberatory installations and their auxiliaries. The one arc furnace can handle the Perth Amboy refinery's entire output on a 5-day operating schedule, compared with the several reverberatory furnaces required to do the same job. Furthermore, reverberatory furnaces have always demanded a large investment in waste heat boilers to make them thermally efficient.

The combination of an oil-fired pre-heating furnace with the arc furnace serves to reduce the power costs, although the overall power costs for the unit in the New Jersey region is higher than for the reverberatory furnace. This is, however, more than matched by other lower operating costs.

The reverberatory furnaces produce a considerable quantity of slag which must be retreated, while the arc furnace produces less than one fourth as much slag. The reverberatory furnace requires expensive iron blow pipes and poles, and furthermore the refractory life in the reverberatory furnace is much shorter.

Finally, and most important, the new melting process lends itself to the new casting technique by providing a continuous stream of molten metal as compared with the batch operation of the reverberatory furnace.

The new casting plant considered in its entirety—arc-furnace, melting and continuous casting—can certainly be considered the first major advance in copper refinery practice in the last 50 years.

1958 JOURNAL OF METALS AWARD PAPER

SMELTING TITANIFEROUS ORES

An old problem reexamined: high titanium ores are here shown to be amenable to modern beneficiation and blast furnace techniques.

by H. U. Ross

TITANIFEROUS ore has been defined[1] as iron ore containing more than 3 pct and less than 25 pct TiO_2, or titania. Ores of this type are found throughout the world, often in very large deposits. Unfortunately most of this ore is left untouched due to the problems met with in smelting it. The objection made to its use by iron smelters is not that the iron made from it is inferior in quality, but that it makes a pasty slag; that it forms accretions, causing scaffolding in the furnace, and clogging in the hearth; and that the amount of fuel required for its reduction is excessive in comparison with that necessary for the reduction of non-titaniferous ores.

On reviewing the literature for data on the smelting of titaniferous ores, it immediately becomes apparent that very little has been written on the subject for many years, at least in the English language. With the development of modern methods of ore beneficiation, whose aim it is to improve the operation of the blast furnace and increase production, it is felt that the time has come to take a critical look at the problems of smelting titaniferous ores with a view to ascertaining whether or not the objections cited can be overcome by up-to-date techniques.

Review of papers written in German, Russian, and other languages indicates that much more study has been carried on, and greater progress made, abroad than in North America. In fact, it has been reported[2] that titaniferous ores have been smelted successfully in Russia for the past fifteen years. It would appear, therefore, that titaniferous ores could be smelted in America equally well provided the proper technique is followed.

Titaniferous iron ores may be smelted for one of several purposes. In one case it may be desired to produce, in addition to pig iron, a titanium-bearing slag from which the titanium could be abstracted as a by-product. In another case it may be desired to make a special grade of pig iron containing small amounts of titanium. In still another case it may be desired to treat the titanium merely as another gangue constituent in the ore to be got rid of in the slag by proper fluxing. Each of these requires its own special technique. The third objective permits the greatest flexibility in choice of method and, unless otherwise stated, it is to be understood that this is the objective aimed for in the following discussion.

H. U. ROSS is Professor of Metallurgical Engineering, University of Toronto, Toronto, Canada. Paper presented at the 1958 Blast Furnace, Coke Oven, and Raw Materials Conference, Cleveland, April 1958.

High fuel consumption

From the small amount of information available, it would appear that one of the major objections to the use of titaniferous ores in the past was the abnormally high consumption of fuel, sometimes as high as three tons for each ton of iron made.

On examination of the ore analysis, it is not hard to visualize one reason why the fuel consumption should be so high. With large concentration of titanium dioxide in the ore, the concentration of iron must be abnormally low. For example when the titanium dioxide runs as high as 25 pct, the iron might be only 40 or 45 pct. This means that an unusually large quantity of ore is required to produce one ton of iron; that greater quantities of fluxes are needed, and that larger slag volumes result. All of these will mean that excessive amounts of fuel will be required to smelt the ore and that furnace capacities will be greatly reduced.

Probably the greater proportion of titaniferous ore deposits are magnetic and lend themselves to magnetic concentration. With modern methods of concentration and agglomeration, it is now possible to consider the use of these ores again after they have been suitably concentrated to raise the iron content and lower the gangue content as much as possible. With straight magnetite ores, it is usually aimed to reduce the silica content to less than 10 pct. If titanium dioxide is considered to be another gangue constituent like silica, then it is logical to attempt to make the sum of the silica and the titania less than 10 pct, if the ore is expected to compete commercially wih other beneficiated ores.

Another factor that probably contributed greatly to the high fuel consumption is the poor reducibility of titaniferous ores. In this respect, these ores are similar to the straight magnetite ores which are hard, dense, and resistant to attack by the reducing gases in the furnace. This results in the well-known phenomenon, *Solution Loss*, with its inevitable high consumption of carbon. Again modern methods of ore beneficiation make it possible to overcome this problem. As with the straight magnetite concentrates, titaniferous magnetites can be sintered or pelletized under the proper oxidizing conditions to convert the Fe_3O_4 to Fe_2O_3, greatly improving the reducibility. Thus it is seen that, provided the titaniferous ore lends itself reasonably well to magnetic concentration, one of the major objections to its use in the past can be largely overcome.

Build-up of hearth accretions

A second objection to the use of titaniferous ore is the build-up of accretions in the hearth of the

Table I. Range of Slag Compositions with Liquidus Temperatures of 1400°C or Lower

Titania, Pct	Lime, Pct	Silica, Pct	$CaO-SiO_2$ Ratio	
			Low	High
10	33-35	57-55	0.529	0.636
20	30-40	50-40	0.600	1.000
30	27-36	43-34	0.628	1.059
40	23-31	37-29	0.622	1.069

blast furnace. One report[3] states that the copper-colored deposits found in the joints of the hearth bottom were identified as titanium carbide and titanium nitride with smaller amounts of other metallic elements. Titanium carbide has a melting point of 3150°C and titanium nitride has a melting point of 2950°C. They have identical crystal structures of the sodium chloride type and form a continuous solid solution in all proportions[4] which might be called titanium *carbonitride*. It can be seen, therefore, that this material will occur in the solid state at all temperatures that may exist in the hearth of a blast furnace, and that unless the hearth is continuously scoured by the flow of molten iron and slag, accretions will build up in the furnace over a period of time, if these compounds are allowed to form.

Titanium carbide is formed by the reaction between titanium metal and carbon according to the reaction: $Ti + C \rightarrow TiC$. This reaction cannot take place if titanium does not exist as the metal. In turn, the metal is produced by the reduction of titanium dioxide by carbon as follows: $TiO_2 + 2C \rightarrow Ti + 2CO$. Titanium dioxide has a high free energy of formation and requires a very high temperature and strongly reducing conditions in order to be converted to metal. Titanium metal cannot exist in the presence of iron oxide; therefore, if the furnace is operated in such a way as to leave a small amount of iron oxide in the slag, no trouble will be experienced from the formation of titanium carbonitride.

Slag viscosity

The third complaint attributed to the use of titaniferous ores is the high slag viscosity which causes trouble in working the furnace, and in handling of the slag, and gives rise to the formation of skulls in the iron ladles. Much of this trouble could be the result of minute crystals of titanium carbonitride entrained in slag and metal increasing the viscosity of the former and skulling the ladles as it segregates from the latter.

Rosenqvist[5] found that, when a titania-bearing slag was melted in a ceramic crucible, it was quite fluid but when the same slag was melted in a graphite crucible, it was fluid at first but thickened over a period of time when held at a high temperature. From this he concluded that the thickening was the result of the formation of lower oxides of titanium and of titanium carbide. Stansfield[6] noted also that titaniferous slags become thick when held in graphite crucibles at temperatures above 1400°C.

Rosenqvist also found on further study that, provided the titania content of the slag did not exceed 15 pct and the titanium content of the metal was kept below a maximum of 0.5 pct, no trouble need be anticipated even under reducing conditions. However, this low titania concentration in the slag does not permit the use of blast-furace burdens containing large quantities of titanium dioxide unless abnormally large slag volumes are permitted. For example, if the slag volume is limited to 1000 lb per ton of pig iron and the iron in the ore is as high as 60 pct, then the titanium dioxide in the burden must not exceed 5 pct, if the titania in the slag is to be limited to 15 pct. This greatly reduces the choice of titaniferous raw materials inasmuch as these ores usually run much higher in titania. In order to use a wider range of ore analyses, it is necessary to use

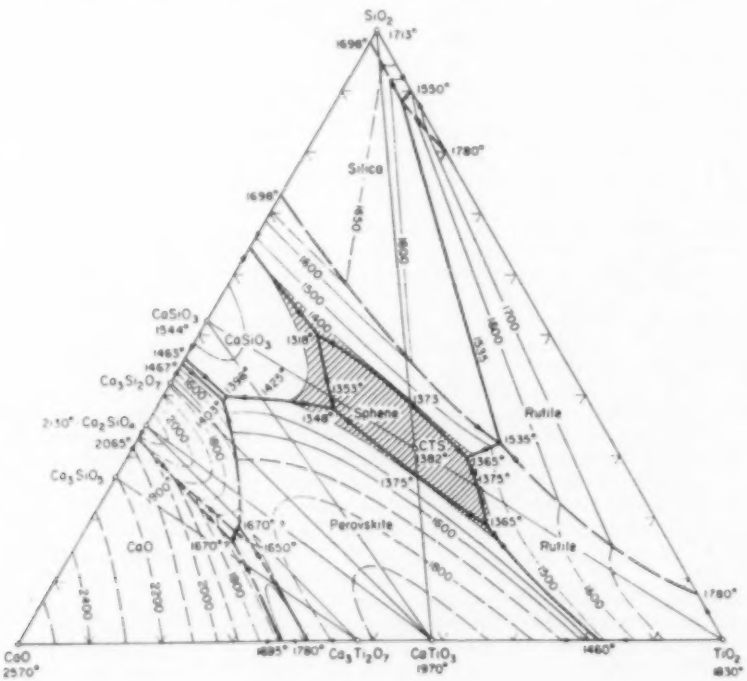


Fig. 1—Phase diagram for the system lime-silica-titania. Shaded area shows compositions of slags liquid above 1400°C.

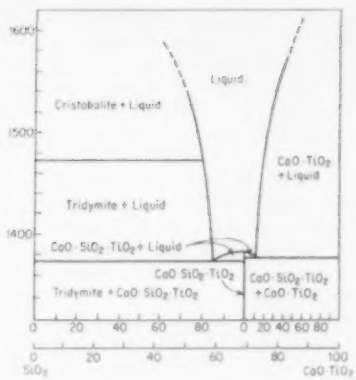


Fig. 2—Phase diagram for system SiO_2-CaO·TiO_2.

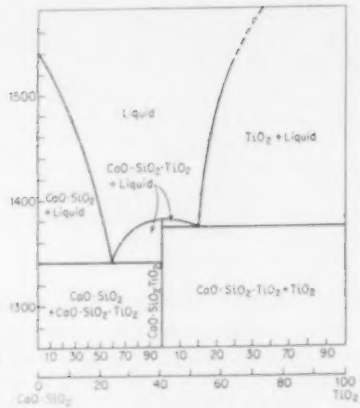


Fig. 3—Phase diagram for system Cao·SiO_2-TiO_2.

slags with a much higher titanium dioxide content, perhaps 30 pct or over.

In order to arrive at the optimum titaniferous slag compositions for trouble-free operation, it is interesting to look at the work done by the Russians. Mikhailov and Belyakova[7] found that titanium dioxide decreased the viscosity of acid slags and increased the viscosity of basic slags; that titanium sesquioxide (Ti_2O_3) and titanium monoxide (TiO) increased the viscosity of both acid and basic slags; and that titanium carbide, which is formed in larger amounts in basic slags than in acid slags, considerably increases the viscosity of both acid and basic slags. They found that ferrous oxide (FeO) decomposed titanium carbide and thus improved the slag fluidity. High temperatures cause the formation of titanium carbide whereas low-temperature smelting facilitates the preservation of ferrous oxide (FeO) and manganese oxide (MnO) in the slag and retards the formation of titanium carbide.

Lyuban and Basov[8] smelted an ore in a blast furnace containing:

52-54 pct Fe
12-14 pct TiO_2
3-5 pct SiO_2
3-6 pct Al_2O_3
3-6 pct MgO
Trace CaO

The slag made contained:

30-35 pct CaO and MgO
27-30 pct TiO_2
17-18 pct SiO_2

The success of smelting titaniferous magnetite was found to depend on the proper ratio of silica to titania in the charge and silica had to be added as a flux to insure the proper ratio.

Semik[9] measured the viscosity of titaniferous slags. In his experiments he added titanium dioxide to a natural blast furnace slag with a lime-to-silica ratio of 1.06. He found that the melting point and the viscosity of the slag increased as the titania was increased up to 30 pct (the limit of this experiments). However, at high temperatures there was very little difference in the viscosities. He then increased the lime-to-silica ratio of the slags to 1.64 by adding lime, keeping the titania around 27 pct. He found the melting point and the viscosity to increase drastically. Finally he made another series of tests in which he lowered the lime-to-silica ratio to 0.68 by adding silica. This lowered the melting point and the viscosity and obtained a slag with characteristics similar to the original natural slag.

It would appear from these observations that the optimum titaniferous slags are those with a low lime-to-silica ratio, that is slags that are definitely acidic in nature. It can be clearly seen in Fig. 1 why this is so. This is the phase diagram for the ternary system lime-silica-titania,[10] with contour lines showing temperature levels. Slags, made up of the three components lime, silica, and titania, whose compositions are such that they are all liquid at temperatures above 1400°C, are shown in the shaded area inside the 1400° contour line. This area shows that there is a very small spread between the limits of the lime-to-silica ratio allowable within this temperature range. Table I shows the limits of lime and silica contents and the lime-to-silica ratios for slags containing 10, 20, 30 and 40 pct TiO_2 whose liquidus temperature is 1400°C or lower. Outside this range of composition, the liquidus temperature rises very quickly to 1600°C and over. Figs. 2 and 3 show how sharply the temperature rises outside the desired limits and how narrow is the range of low-temperature melting slags.

Titaniferous ores, in general, particularly concentrates, are quite low in silica. Thus in order to make slags of the desired composition, it is usually necessary to add a silica flux as well as lime. The silica could be added as sand or gravel, but it is probably more advantageous to add a siliceous iron ore which is usually readily obtainable.

For example, suppose a titaniferous ore and a siliceous ore have the following composition:

Component	Titaniferous Ore, Pct	Siliceous Ore, Pct
Iron	55.95	45.55
Silica	2.55	16.54
Manganese	.18	.75
Alumina	3.98	2.03
Lime	.12	.19
Magnesia	1.62	.13
Titania	11.80	—

The titaniferous ore is a typical concentrate that might be produced from an Adirondacks magnetite, while the other is a high siliceous ore from the Mesabi Range. If these two ores were smelted in equal proportions with 1800 lb of coke and 675 lb of limestone, it can be shown that there will be produced about 1320 lb of slag of the following composition:

Component	Slag Analysis, Pct
Silica	34.2
Manganese	.9
Alumina	14.3
Lime	27.2
Magnesia	6.1
Titania	17.3
	100.0

The lime-to-silica ratio is 0.797. If only the three major components of the slag are considered, the composition would be

Silica	43.5 pct
Lime	34.6 pct
Titania	21.9 pct

This shows that the slag composition follows within the required limits to obtain a low liquidus temperature.

Importance of low liquidus temperatures

It is important to produce a slag with a low liquidus temperature. If a slag is of incorrect composition and, if it is attempted to make it more fluid by running the furnace hotter, the result will be that titanium metal will be reduced, and titanium carbide and nitride will be formed. These will only tend to make the slag more viscous. Eventually, accretions will begin to build up in the hearth and troubles will develop. On the other hand, with a slag of the proper composition, it will be quite fluid even when the furnace is relatively cold. This appears to be the secret to the successful smelting of titaniferous ores.

Effect of other slag components

So far only the three major components—silica, lime, and titania—have been considered in this discussion. When other components are added to these three, the slag becomes more complex and it is difficult to visualize the changes that take place when small amounts of minor components are added. Undoubtedly, however, minor constituents, such as alumina, magnesia, iron oxide, manganese oxide, and alkalis play an important part in the production of suitable slags.

The slag with the lowest melting point in the three-component system has a composition of about 50 pct SiO_2, 33 pct CaO, and 17 pct TiO_2. The melting point of this eutectic is 1318°C. With four or more components, it is possible to obtain slags with even lower melting points. For example, there is a eutectic at 1245°C in the four-component system containing about 43.2 pct SiO_2, 32.7 pct CaO, 10.2 pct TiO_2, and 13.9 pct Al_2O_3, and there is a wide range of compositions containing these four components in which the liquidus temperature is below 1350°C. Thus, it may be concluded that alumina in the proportions usually found in blast-furnace slags is highly beneficial. Magnesia, iron oxide, and manganese oxide have a similar effect on the lowering of the melting point of the slag. In addition, the presence of the oxides of iron and manganese in the slag retard the reduction of titanium, which is most desirable if the formation of titanium carbide and nitride is to be avoided.

The effect of alkalis, even in very small concentrations, is worthy of note. Here again it is interesting to see what the Russians have done.[12] When smelting titaniferous magnetite with low-sulfur Siberian coke, it became established that the optimum lime-to-silica ratio was 1.0 to 1.1. However, with the use of coke containing more sulfur, it became necessary to develop conditions under which the smelting operation could be conducted to yield a more basic slag. With this object in view, a test on a blast furnace was run in which miaskite, a low-melting alumino-silicate mineral containing 13.5 to 15 pct alkali, was added in the charge at the rate of about 270 lb per ton of pig iron. This resulted in a concentration of about 2 pct alkali in the slag and produced a stable titaniferous silicate slag with 14 to 16 pct TiO_2 and a lime-to-silica ratio of 1.2 to 1.4. Slags with this basicity showed a good desulfurizing capacity.

Halversen[13] has patented the use of sodium chloride in the smelting of titaniferous ores. In his process, a small quantity of salt is injected into the furnace through the tuyeres in sufficient amount to give a concentration of 1 to 2 pct soda in the slag. This produces a very fluid slag even at low temperatures which flows like water. It is claimed that the soda prevents the reduction of titanium and the formation of the objectionable carbide and nitride. However, it is difficult to visualize how the soda could accomplish this, and it is more likely that the reduction of titanium is prevented due to the low temperature at which it is possible to run the furnace because of the free-running nature of the slag.

Halversen did not carry his experiments far enough to show whether there would be any build-up in the furnace hearth or whether the sodium chloride had any damaging effect upon the furnace refractories; it is feared that any advantage obtained through the use of salt might be offset by the disadvantage caused by the deterioration of the refractories.

Desulfurization of hot metal

Although the use of an acid slag with a low lime-to-silica ratio and a low hearth temperature apparently solve the problem of viscous titaniferous slags, these factors introduce a new problem in the high sulfur content of the hot metal that will result. The modern well known blast-furnace process uses a basic slag and high temperature in order that sulfur will be absorbed primarily in the slag rather than in the iron. However, titaniferous ores require a departure from conventional procedure and some other means for desulfurization is required. A few years ago this problem alone would have been sufficient to prohibit the use of these ores, but with modern methods of desulfurization in the ladle outside the furnace, several tested possibilities are available as a solution to the problem.

At the 1956 Conference of the Blast Furnace, Coke Ovens, and Raw Materials Committee, several papers were presented on the desulfurization of hot metal. Elliott[14] discussed the theoretical principles of sulfur removal and showed that high carbon and silicon concentrations in the metal were beneficial in aiding the process. Watkins and McFarland[15] discussed the use of calcium carbide as a reagent; Hornak and Whittenberg[16] compared the injection of strong alkalis, lime, calcium carbide, and calcium cyanamid; while Hatch, Wagner, and Trottier[16] described the use of calcium cyanamid for desulfurization of iron in ladles. It is interesting to note that the case described by Hatch is one where titanium ores are smeltd in electric furnaces to produce a 70 pct titanium-bearing slag and a pig iron byproduct and illustrates clearly the problem of desulfurization when smelting titaniferous ores.

Acid smelting followed by external desulfurization is not new. It has been practiced for years at Corby, England where the sulfur in the iron is lowered from 0.25 to 0.045 pct using soda ash as the desulfurizing reagent. It is reported[17] also that the process is in use in at least one furnace in Germany

where iron containing up to 0.21 pct S is treated successfully externally.

Hornak and Whittenberger point out that external treatment appears attractive in that, by this approach, the furnace can be burdened for maximum production and minimum operating cost, and that the cost of external treatment must be economical as compared with the cost of burdening the furnace. This applies also to titaniferous ores and, when calculating the cost of producing iron from these ores, the cost of external desulfurization must be taken into account.

Reduction of silicon

The reduction of silicon in the presence of large concentrations of titanium dioxide without reducing titanium presents a difficult problem. The free-energies of formation of silica and titania are very close together.[18] Both silicon and titanium require strongly reducing conditions and high temperatures for their reduction. A rise in furnace temperature will increase the concentration of both elements in the pig iron unless controlled by other variables, such as the lime-to-silica ratio, and the titania content of the slag. As explained in a previous section, no trouble from the precipitation of titanium carbonitride need be feared provided the titanium in the pig iron is kept under 0.5 pct. The problem of silicon reduction resolves itself into one of controlling the silicon at the level specified without allowing the titanium to exceed this allowable maximum. The work of Wentrup, Maetz and Heller[19] indicates how the silicon might be controlled under these conditions. They found the lime-to-silica ratio to be very important in that, as compared to silicon, titanium is reduced more easily from basic slags than from acid slags. In other words, as the lime-to-silica ratio is decreased, the silicon reduction is increased for the same amount of titanium reduction. In this way it is possible to produce pig iron containing up to about 2 pct Si without exceeding 0.5 pct Ti.

The influence of temperature upon the reduction of silicon and titanium is interesting. In very acid slags, rising temperatures are accompanied by an increase in silicon reduction only with practically no effect upon the titanium. On the other hand, in more basic slags, the titanium reduction is also enhanced.

The titania content of the slag also affects the relative amounts of silicon and titanium reduced. As the titania in the slag increases, the titanium in the iron increases while the silicon in the iron decreases. This appears to be simply the result of an increase in the activity of the titanium and a decrease in the activity of silicon caused by their respective changes in concentration in the slag. It may be concluded from this that the silica-to-titania ratio, as well as the lime-to-silica, is very important in controlling the desired silicon content of the pig iron. By controlling these two variables and the temperature, it is probable that pig iron containing up to 2 pct silicon might be made successfully from titaniferous ores but for higher silicon concentrations, trouble might be expected.

Summary

From the results of the studies that were made on the smelting of titaniferous ores, the following conclusions may be drawn:

1) The high fuel consumption normally associated with the smelting of titaniferous ores can be reduced if the ore is amenable to magnetic concentration and beneficiation by pelletizing.

2) Both the build-up of hearth accretions and the formation of viscous slags can be prevented by proper temperature and slag control. This involves reduction at a low temperature and the use of an acidic slag with a low melting point and a very low lime-to-silica ratio. The low melting point of the slag is essential in order that the slag will be completely fluid at the low hearth temperature employed.

3) The use of an acid slag process necessitates that the hot metal made be desulfurized by some external treatment outside the furnace. Soda ash, lime, calcium carbide, and calcium cyanamid have been proved to be efficient desulfurizing reagents.

4) There are indications that the addition of alkalis to the slag in small amounts, either by the use of suitable fluxes or through sodium chloride injection through the tuyeres, will enable the use of more basic slags and will permit the desulfurization of the hot metal by conventional methods.

5) By proper control of the lime-to-silica ratio, the silica-to-titania ratio and the temperature, it is possible to control the silicon content of the pig iron to any specified amount up to about 2 pct without permitting the titanium content to exceed the maximum allowable limit of about 0.5 pct.

In general, it may be said that the smelting of titaniferous ores has been greatly simplified by recent developments in ore beneficiation and external desulfurization of hot metal. With these modern techniques, coupled with the slag and temperature control best suited to the ore at hand, it may be concluded that the smelting of titaniferous ores is a technical possibility. Furthermore, under favorable conditions in supplying the raw materials to the blast furnace, the treatment of these ores is economically feasible.

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SUSPENDED BASIC ROOFS FOR COPPER FURNACES

1. At Canadian Copper Refiners Ltd.

by Guy Bridgstock

THE anode furnace at Canadian Copper Refiners Ltd.'s Montreal East Refinery converts blister copper, tank-house anode scrap, and scrap copper into anodes for electrolysis in the tank house. In addition, silver refinery slags and any metalbearing reverts from other sections of the plant are treated, thereby imposing severe service on the furnace.

The original 150-ton anode furnace had a silica roof, walls, and bottom. From 1931 to 1945 it was normal to replace the roof and sidewalls yearly and the bottom at four to five year intervals. The use of silica brick for sidewalls was changed over to basic brick in 1942, and in 1945 the construction of the roof was modified with a suspended basic-brick hood and verb being installed.

With this form of roof construction, eight roofs were installed in the period March 1945 to April 1953. The cost of installing a new silica roof in 1951 was $9050, of which $6810 was for refractories, and $2240 for labor.

Repairs were made to the silica roofs during their life, the majority being to the verb, hood, and uptake. Generally, the first repair was made to the hood after approximately 90 charges. Minor repairs could be made to the main roof, but they were very difficult to carry out.

Change to basic roof

With only one anode furnace, shutdowns to replace roofs were inconvenient and costly, since the average operating time lost for a roof rebuild was 12 days. Therefore, a complete suspended roof of Detrick design, shown below, was installed on the furnace in May 1953. The whole operation took 21 days from the time the furnace was emptied until the first charge was cast. At this same time the entire furnace, with the exception of the silica bottom, was replaced. The cost of installing the basic roof was $20,310, made up of $14,490 for refractories, $3500 for steel, and $2320 for labor.

The shape of the roof arch was obtained by using radial suspension. This provides a smoother roof surface and enables uniform length brick suspension hooks to be used. A further advantage is that the arched shape tends to prevent the loss of any large pieces of brick by spalling.

A few minor changes were needed to the existing furnace steel structure so that the roof could be installed. The I-beam frame-work to support the suspended roof was connected to the buck stays, and spring loaded supports for the skewbacks were installed.

The basic suspended brick construction extends from the fire-box roof, which is still silica, to the top of the uptake where it connects with the waste-heat boiler brickwork. Originally 75 pct of the basic roof verb, hood, and uptake was built with Ritex 40, a General Refractories brick. The remaining 25 pct of the roof close to the fire box was built with Brucite, a Canadian Refractories' brick. Both are chemically bonded magnesite-chrome refractories, containing approximately 60 pct magnesite-bearing materials. These two types of brick were installed in this first basic roof so that a comparison could be made under operating conditions. It was found that both types of brick gave equally good service.

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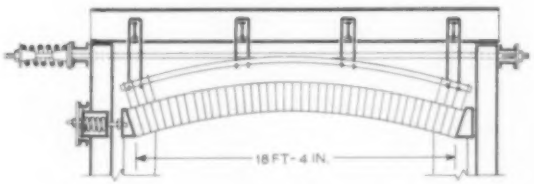

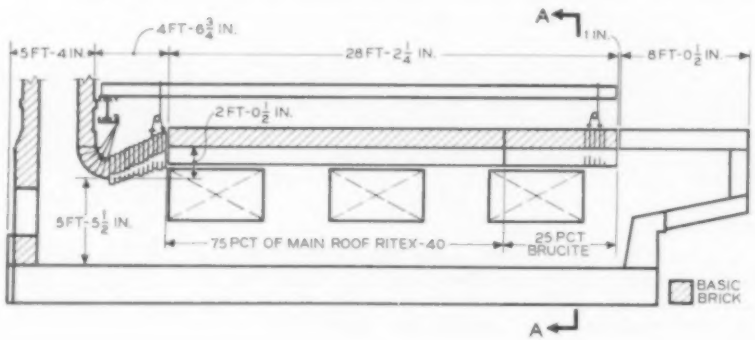

Suspended basic roof design in use at the Montreal East refinery of Canadian Copper Refiners Ltd. At right the profile is shown, and above is a cross-section at section A-A. Installation of the new unit took a total of 21 days.

Operating results

The furnace operated with this suspended roof for 35 months, during which time 878 charges were cast for the production of 225,812 tons of copper. However, the furnace was shut down after 496 charges to replace the sidewalls and bottom. Approximately 75 pct of the main portion of the roof was left intact during the repair operations.

Minor repairs were made to the main roof, hood, verb, and uptake during their life. These repairs consisted largely of patches, but they were considerably easier to carry out than with the silica roofs formerly used.

The second basic roof installed in May 1956 was built exclusively with Brucite. The first repair to this new roof was made in December 1956, after 116 charges had been cast. On May 21, 1957, after 319 charges had been cast, the furnace was shut down to rebuild the sidewalls. The hood was rebuilt, and one course of brick, next to the skewbacks, was replaced. To date, three other small repairs have been made to the verb and one to the main roof.

Conclusions

The life obtained from the first suspended basic roof installed on the cold-charged anode furnace at Canadian Copper Refiners has been three times that previously obtained with a silica roof and has increased the availability of the furnace. For 35 months' operation with basic roofs 878 charges were cast, compared with 785 charges for the silica roof during a comparable period.

A comparison of costs for the anode furnace with a silica roof and basic roof are shown in Table I. These include the cost of installing new roofs and all normal operating repairs to the furnace. The cost for sidewall and bottom replacements have not been included, but the costs for labor and material have been corrected to conform with rates and prices prevailing in 1956. These cost figures show that the increased life obtained from a basic roof has reduced the roof repair costs for the anode furnace by 25 pct.

Table I. Comparison of Roof Repair Costs at Canadian Copper Refiners Ltd.

	Silica Roof		Basic Roof	
	Cost, $	Manhours	Cost, $	Manhours
1st Year				
New roof	11,590		20,310	
Repair	18,795	1897	5,183	918
2nd Year				
New roof	11,590		—	
Repair	23,452	2410	14,920	2017
3rd Year				
New roof	11,590		—	
Repair	11,700	1138	11,569	1547
4th Year				
New roof	11,590		22,893	
Total	100,307	5445	74,875	4482

2. At United States Metals Refining Co.

by John Towers and Robert N. Brown

Attempts to prolong furnace campaigns and also reduce brick repair costs, have prompted progressive changes in roof practice on reverberatory refining furnaces at Carteret, N. J. One of the 340-ton capacity furnaces was equipped and operated with several types of roofs, including sprung silica arches, silica-basic zebra construction, and finally an all-basic design. This furnace, 13.2 ft wide by 43.5 ft long, may be regarded as fairly typical of the copper refinery reverberatory. The operations involved in charging, melting, refining, and casting the routine electro wirebar charge are carried out on a 24-hr cycle.

The all-basic roof

In 1955, No. 8 furnace was completely rebuilt, and a radially suspended basic roof installed. The roof was designed by M. H. Detrick Co., using 15-in. steel-clad, chemically-bonded chrome-magnesite brick furnished with internal metal plates. Supporting steelwork for the suspension was provided by spanning opposite buckstays with 18-in. I-beams.

The furnace has operated since August, 1955 with but one shutdown for roof repair. Whereas it had been anticipated that an all-basic roof would yield 300 to 350 charges and then require 50 pct replacement, the roof actually remained in service for 471 charges prior to requiring repair. It would have been possible to prolong the furnace campaign but for the failure of water-cooled skewbacks. Replacements of the latter members necessitated a shutdown during which it was decided to repair a section of the roof which was comparatively thin—4 to 6 in. The portion of the roof replaced represented 34 pct of the total suspension. The balance of the roof left intact ranges from 8 to 12 in. in thickness.

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Comparison of results

A comparison of silica, zebra, and suspended basic roofs used on No. 8 wirebar furnace is shown in Table II. Although the furnace was enlarged coincident with installation of the suspended roof, the unit costs given are calculated on the basis of 300 tons per charge in order to provide a direct and fair comparison.

Installation of the suspended basic roof has resulted in a decrease in furnace shutdowns required for roof repair. Avoidance of such repairs permits uninterrupted production programs and has, thus, alleviated the costly necessity of starting up a small capacity spare furnace. Further, we are satisfied that, in spite of its high initial cost, the all-basic roof has proved to be more economical in operation than other types of construction previously employed on Carteret wirebar refining furnaces.

Table II. Comparison of Different Type Roof Construction at Carteret, N. J.

	Silica Construction	Zebra Construction	Suspended Basic Construction
Cost per ton of copper cast, $	0.261	0.216	0.168
Avg no. of charges per roof	142	159	583
Cost of new roof, $	8,490	10,345	25,145
Avg no. shutdowns per year for roof rebuild or repair	2-1/3	1-1/2	<1

DIRECT ELECTROREFINING OF NICKEL MATTE

International Nickel Company has unveiled a new electrorefining process which recovers high purity sulfur, selenium, and cobalt.

by L. S. Renzoni, R. C. McQuire, and W. V. Barker

A RECENT development (U. S. and Canadian patents pending) now enables the International Nickel Co. of Canada, Ltd., to recover elemental sulfur and selenium, as well as high purity cobalt metal, during the course of nickel electrorefining. The new process eliminates traditional operations of high-temperature sintering of bessemer matte and reduction of the oxide to metal anodes for electrorefining. Instead, the molten matte is cast directly into sulfide anodes and electrolyzed.

History

The history of nickel electrorefining is replete with trials and disappointments. This was particularly true at the turn of the century, when there were many proposals for the treatment of materials ranging from raw ores to metal anodes in a variety of electrolytes. with the advent of the Hybinette[1] process, the industry settled down to an extended period of refining crude metal anodes in a sulfate-boric acid electrolyte. Although chloride-bearing electrolytes had been used in the electroplating industry since 1915, and several major advantages were visualized for such an electrolyte in refining, the attendant serious problems prevented development of a successful process until 1942.

In 1942, following five years of development, Inco converted to a sulfate-chloride electrolyte.[2,3,4] This revision permitted the use of higher current densities with consequent increase in plant capacity and significant power saving. It laid the foundation for the recovery of cobalt as a valuable by-product and also was a factor in the sulfide anode development.

Metal anode operation

During nickel refining with metal anodes, the sulfur content determines, in large measure, the quantity of anode slime; so, sulfur is kept at a low level. Inco metal anodes contain about 0.7 pct S, but some others have used higher sulfur contents; Brown[6] attempted to use anodes containing 14 pct S, while Hybinette,[5] Giertsen,[7] and Bryk[8] taught the use of anodes containing up to 10 pct S. However, all of these behave as metal anodes. Upon anodic corrosion, the metallic phase dissolves selectively, leaving the sulfides undissolved. With anodes containing 10 pct S, the anode residue would be about 30 pct of the original anode weight. This, together with about 20 pct scrap, would have to be reprocessed.

Avoidance of the various high-temperature operations required for the production of metal anodes has long been desired. Interesting possibilities were first demonstrated at Inco in 1951, when a small piece of cast nickel sulfide was electrolyzed. It corroded smoothly to dissolve nickel and other base metals, leaving an anode sludge of elemental sulfur, the precious metals, and only small amounts of undissolved sulfides.

Laboratory studies

Laboratory studies which followed were carried out with refinery electrolyte, using 25 pct S anodes, 0.5 in. thick, which had an effective area of 48 sq in.

The mechanism of sulfide anode corrosion differs from that of metal anodes whereby nickel and other metals are oxidized to the ionic state. With sulfide anodes, the metals are already essentially in the ionic state, and the anodic reaction involves oxidation of sulfide sulfur to the elemental state, with release of the metal ions to the solution. This reaction, $Ni_xS_z - 6e \rightarrow 3Ni^{++} + 2S$, requires an increase in anode potential from 0.2 v for metal anodes to about 1.2 v for sulfide anodes. The anode sludge contains about 95 pct elemental S and 0.15 pct elemental Se, with the remainder precious metals and loose sulfides.

Another significant observation was concerned with anode efficiency. It was found that sulfide anodes corroded at 95 pct current efficiency, as com-

L. S. RENZONI, R. C. McQUIRE, and W. V. BARKER are Supt. of Research; Mgr., Nickel Refining Div., Port Colborne, Ont.; and Chief Research Chemist, Nickel Refining Div., respectively, International Nickel Co. of Canada, Ltd. Paper presented at AIME Annual Meeting, New York, February 1958.

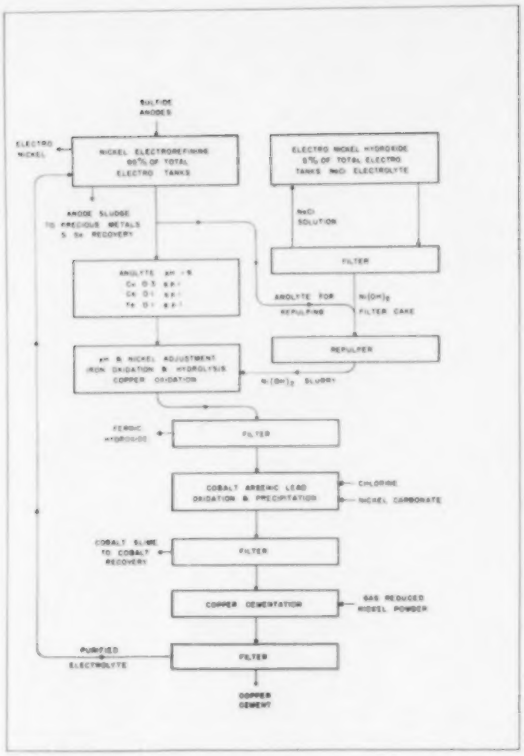

Fig. 1—Flowsheet for producing electrolytic nickel from sulfide anodes.

pared with 99.5 pct for metal anodes. The remaining 5 pct of the anodic current was consumed for hydroxyl ion discharge with consequent evolution of oxygen and increase in anolyte acidity. There was no chloride ion discharge, and no significant oxidation of sulfur was detected.

Attempts were made to increase anode efficiency by addition of sodium fluoride, ammonium chloride, and sodium fluorborate to the electrolyte, but without success. Similarly, minor additions to the sulfide failed to provide significant benefit.

Results of exploratory tests to determine the effect of electrolyte composition on anode activity indicated an optimum chloride content within the range of the existing refinery electrolyte. An all-sulfate electrolyte was relatively unattractive, because of the low anode activity and the low sulfur content of the anode sludge. Surprisingly, an all-chloride electrolyte also proved less desirable than the sulfate-chloride solution.

Other tests were concerned with the effects of varying the sulfur content of the anodes. Results indicated that the sulfur content should be in excess of 23 pct for highly satisfactory performance. At upwards of 25 pct S, the anodes corrode at a cell voltage of 2.3 v to produce an anode sludge containing about 97 pct elemental S. With a sulfur content below about 25 pct, a metallic phase separates during freezing and such metallics dissolve selectively during electrolysis, leaving a network of uncorroded nickel sulfide. However, this does not become significant until the sulfur falls below about 23 pct and is not serious until sulfur content falls below 20 pct. Thus an anode containing less than 20 pct S corrodes largely as a metal anode at a cell voltage of 1.7 v to produce a hard sludge containing only 50 pct elemental S. Increasing anode sulfur content up to 32 pct showed no deleterious effects.

Copper is a major impurity in nickel refining but has only a minor effect on the corrosion of sulfide anodes, and that only in determining the sulfur content necessary to avoid its selective solution. In the normal course of matte production, nickel and copper occur mainly as Ni_3S_2 and Cu_2S. However, in sulfide anodes, Cu_2S was shown to dissolve selectively in a manner similar to metallics. The effect of this selective solution is not a serious factor until the copper content exceeds about 10 pct. Above this level, with copper present as Cu_2S, cell voltage drops to about 1.8 v, and the anode sludge becomes a hard network of uncorroded nickel sulfide with low elemental sulfur content.

Tests with a Cu_2S anode indicate that corrosion occurs by the following mechanism: $Cu_2S - e \rightarrow Cu^+ + CuS$. One atom of copper is dissolved at about 0.2 v anode potential, leaving an anode sludge of fine cupric sulfide. However, with sufficient sulfur in the anode, it appears that the copper sulfide then corrodes under conditions closely similar to those for Ni_3S_2, and the two reactions proceed simultaneously.

After almost complete corrosion, anode sludge occupies about twice the volume of the original sulfide anode and is strongly adherent. In contrast, metal anodes have relatively little sludge, and about 50 pct of it falls to the bottom of the tank.

Sulfide anode sludge, washed and dried, contains 97 pct elemental S. Its recovery by volatilization, solvent extraction, or filtration, leaves a residue containing uncorroded sulfides and the precious metals. The ratio of concentration of precious metals is about 125 to 1, as compared with about 20 to 1 for metal anodes.

The progressively increasing thickness of anode sludge during electrolysis is reflected in increasing electrical resistance. Thus, the cell voltage, which is about 2.3 v on the first day, gradually rises to about 4 v near the end of the life of a 2-in. thick anode. No serious problems developed in growing cathodes of satisfactory purity with the new anodes.

Anode production

Before proceeding to larger scale refining, it was necessary to investigate production techniques for full-size anodes. During preparation of laboratory sulfide anodes, a faint crackling sound was observed during air cooling. This was associated with microscopic, transcrystalline cracks, which developed on cooling through the phase transformation range. Though relatively brittle, these anodes appeared sufficiently strong to withstand handling in commercial practice. However, full-size anodes, cast into open horizontal molds, cracked and broke upon air cooling.

Early trials involving use of metal reinforcing strips cast into the anodes, production of anodes by compacting sulfide fines, and electrolysis of granular sulfide contained in bags, were discouraging. On the other hand, studies progressing simultaneously indicated that the cracking could be avoided by slowly cooling the cast anodes.

Cooling rate studies showed that there is an arrest in the curve between 980° and 940°F coincident with phase transformations. The castings were physically sound at temperatures above about 975°F, but when allowed to cool in air, gave off cracking noises at about 500°F. Some hours later, at a lower

temperature, the anodes fractured into several pieces. Optimum anode cooling rates were found to be as follows: mold cooling in air to about 1000°F, followed by controlled cooling to 800°F at 20° per hr, then to about 300°F at 150° per hr, and air cooling. Down to 1000°F the anodes are strong and withstand rough handling. The fully cooled anodes are somewhat fragile and require more careful handling than metal anodes.

There was laboratory evidence that increasing the sulfur content to about 30 pct diminished the effect of the phase changes, but practical difficulties in holding such a high sulfur content discouraged its use.

For the commercial operation, a continuous sulfide melting and anode casting system was developed, featuring semi-automatic mechanized handling essential to the production of satisfactory anodes.

Fine nickel sulfide from the Inco matte separation process[10] is melted in an oil-fired furnace, which has a capacity of 200,000 lb of sulfide per day. Slag, amounting to only 3000 lb per day, is skimmed periodically and reverted to the smelter. Continuous side feeding of sulfide to the furnace is used to maintain a steady melting rate.

Furnace tapping is almost continuous. About 7000 lb of molten sulfide at 1800°F is delivered to a self-propelled hot ladle car on standard gage track. The car moves down a line of 14 horizontal cast iron molds. As it pauses at each, about 475 lb is poured from the hydraulically tilted ladle onto an attached diffuser to spread the sulfide over the mold.

To facilitate anode handling and to provide electrical contacts, two steel anode suspension straps are placed in the mold so that the sulfide is cast around the straps which project about 2 in. into the body of the anode. Dimensions of the anode are 28.5x43.5x1.75 in. The anodes are allowed to cool in the molds to 1000°F and then promptly placed in a controlled cooling box. These boxes have an insulated lining and hold 14 anodes. The insulation maintains the desired cooling rate.

Pilot plant development

Success in anode production allowed transfer of development work to an electrolytic pilot plant, embodying ten full-size electrorefining tanks and a complete electrolyte purification system. Plant capacity was about 2 tpd of refined nickel.

Pilot plant studies were concerned with anode corrosion, electrolyte purification, and production of high-purity cathodes of good appearance. Development of satisfactory tank linings, cathode diaphragms, and anode bags were key problems.

Corrosion of the nickel sulfide anodes was smooth and uniform, superior in this respect to conventional metal anodes. The anode sludge, which was largely elemental sulfur, was granular, porous, and clung to the uncorroded portion of the anode. These factors permitted corrosion of the anodes to about 10 pct scrap, provided that uniform anode weight and good current distribution could be maintained.

In the pilot plant, anode corrosion closely paralleled laboratory experience. The increasing voltage as corrosion proceeded was reflected in increasing heat input to the electrolyte, which entered the tanks at 132°F and left at 145°F. The higher temperature of the anolyte decreased steam requirements in the electrolyte purification section. Anode current efficiency was 95 pct.

Conventional nickel electrorefining tanks are constructed of reinforced concrete, protected by a layer of asphalt. The combination of high temperature and low pH contributed to rapid failure of such lining in the pilot plant tanks. The problem was resolved by the development, in cooperation with the supplier, of a satisfactory synthetic resin lining.

In addition to its effect on the tank lining, anolyte of such acidity seriously affected cathode diaphragms, which lasted only a few weeks. Suitable diaphragm fabrics of dynel and of terylene, each with a special finish, have now been produced.

Collection of the voluminous sludge, which forms during anode corrosion, was facilitated by bagging of the sulfide anodes in leno-weave dynel, with allowance made for an increase in sludge volume. Dynel gives satisfactory service life, which is improved by application of neoprene after weaving.

Electrolyte purification

As stated earlier, corrosion of the nickel sulfide anodes in sulfate-chloride electrolyte takes place at an anode current efficiency of about 95 pct, while the cathode current efficiency is 99.5 pct. Thus, a nickel deficiency in the anolyte arises and is mani-

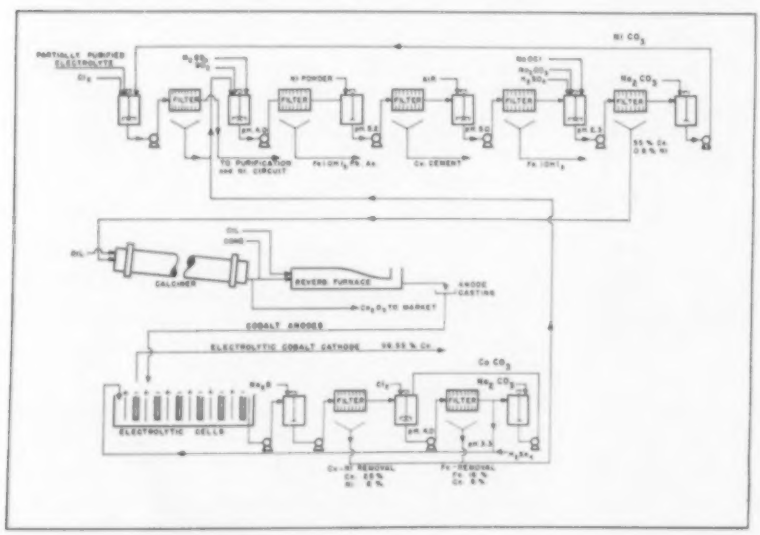

Fig. 2—Flowsheet for the production of electrolytic cobalt.

fested by an equivalent increase in acid content which lowers the pH to about 1.9. The first step in electrolyte purification, therefore, involves the addition of basic nickel material to correct the nickel deficiency and restore the pH to 4.0. This is followed by oxidation and hydrolysis of iron, selective oxidation and hydrolysis of the cobalt, arsenic, and lead, and finally, cementation of copper.

The required basic nickel material can readily be produced as nickel hydroxide by electrolysis of sulfide anodes in a sodium chloride solution. For this purpose about 5 pct of the tanks in the plant are needed. Such tanks contain the usual number of bagged anodes opposite cathodes of light nickel or steel sheet. No cathode compartments are required. The tanks are fed continuously with a solution containing about 50 gpl NaCl. As the anodes corrode, nickel passing into solution hydrolyzes immediately. The usual anode sludge of high elemental sulfur content forms but remains in place. Hydrogen ion discharge at the cathode maintains the required concentration of hydroxyl ions in the solution. Tank voltage is about 3 v. Nickel hydroxide slurry overflows from the tank at about 100°F and pH 8.3 to a filter. The filtrate returns to the circuit after adjustment of its pH and salt content.

Another source of basic nickel salts is available to the refinery as a by-product of Inco's iron ore recovery process.

Commercial-scale operation

From the standpoint of cathode purity, the high quality of nickel produced from metal anodes is maintained in production from sulfide anodes by virtue of rigorous electrolyte purification. Furthermore, in addition to precious metals and cobalt recovery of past practice, commercial recovery of elemental sulfur and selenium is now effected for the first time in nickel refining.

Electrolytic cobalt

Separation of cobalt from nickel electrolyte and production of cobalt oxide have been practiced at the refinery since 1947. Production of high-purity electrolytic cobalt began in 1954.

The process of cobalt recovery in electrorefining of nickel and the purification of cobalt precipitates have already been disclosed.[6] As indicated in Fig. 2, cobalt dissolved from the anodes is precipitated from the nickel electrolyte by selective oxidation with chlorine, followed by hydrolysis to cobaltic hydroxide. Basic nickel carbonate, returning mainly from the cobalt purification circuit, is used for pH control and for replacing cobalt removed from the electrolyte. The primary cobalt precipitate contains large amounts of copper, nickel, and iron, as well as minor amounts of lead and arsenic. This precipitate is slurried with water, selectively reduced with sulfur dioxide, and dissolved with acid. In the resulting solution, nickel and cobalt are in the reduced state, whereas the other elements are only partly reduced. A further quantity of the primary cobalt precipitate is added to raise pH to 4.3 and to reoxidize and reprecipitate the iron, lead, and arsenic. Upon filtration, a solution essentially free of iron, arsenic, and lead, and containing about 25 gpl Co, 50 gpl Ni plus Cu is obtained. The copper is removed by cementation with nickel powder, and the resulting solution is treated with hypochlorite for cobalt-nickel separation.

The cobalt is recovered as cobaltic hydroxide and calcined to oxide, which may be marketed as such or treated further for the production of electrolytic cobalt. The nickel is precipitated with soda ash as basic nickel carbonate for return to the primary cobalt precipitation stage.

The hypochlorite required for cobalt-nickel separation is produced on demand under continuous automatic control by addition of chlorine to soda ash solution in the ratio of 1 lb of chlorine to 3.90 lb of soda ash. This solution has a slight excess of chlorine over the theoretical requirement. Hence, the pH rises to the equilibrium pH 2.2 for cobalt hydrolysis and remains there, while the precipitation proceeds in accordance with the following reaction:

$$2\,CoSO_4 + Cl_2 + 3\,Na_2CO_3 + 3\,H_2O \rightarrow 2\,Co(OH)_3 + 2\,NaCl + 2\,Na_2SO_4 + CO_2.$$

When the cobalt concentration has fallen from 25 to 0.2 gpl, the pH rises sharply to 3.0, and hypochlorite addition is stopped.

Cobalt oxide from these operations is the raw material for the electrocobalt operations. It is mixed with 17 pct petroleum coke, reduced, and melted in an oil-fired reverberatory furnace together with

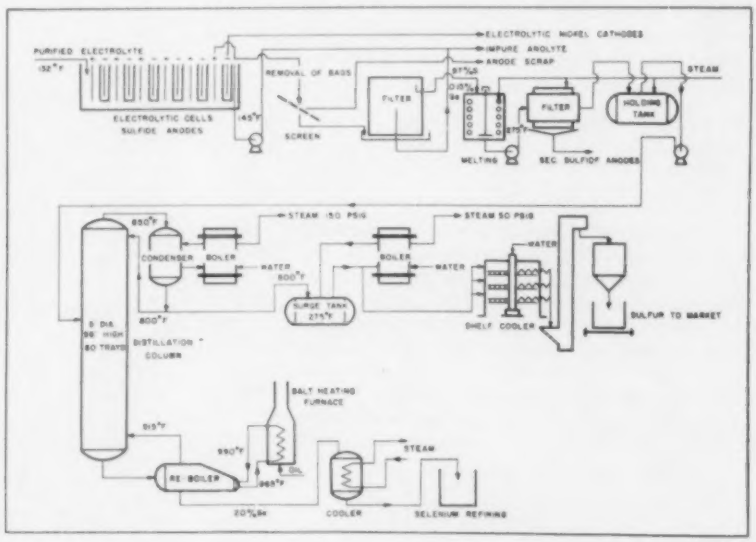

Fig. 3—Flowsheet for the separation of sulfur and selenium.

anode scrap and slime returning from the electrocobalt refining tanks, and cast into anodes similar in size to nickel anodes.

Cobalt anodes are refined in divided cells, the whole plant and process being analogous to—but differing in important details from—electronickel refining. Cobalt anodes as cast have a surface passivity, even in sulfate-chloride electrolyte, and require preliminary anodic pickling in a sulfuric acid-sodium chloride bath. This practice also provides cobalt solution for electrolyte make-up.

The electrolyte contains 60 gpl Co, 40 gpl Na, 170 gpl sulfate, 10 gpl chloride, and 10 gpl boric acid. The principal impurity is nickel, which is held below 2.0 gpl. Electrolyte purification involves precipitation of copper and nickel with sodium sulfide. Iron is removed by oxidation with chlorine at 500 mv redox potential on the platinum-saturated calomel cell, while cobalt carbonate additions maintain pH 4.0. All precipitates revert to the cobalt recovery section of the nickel refinery. Purified cobalt electrolyte is supplied to the cathode compartments at pH 3.3 and 140°F. Electrolysis proceeds at 15 amps per sq ft.

The cobalt deposits on stainless steel plates to a thickness of about 0.10 in. in 7 days. It is stripped off, acid and water washed, dried, and sheared for market. A typical analysis is shown in Table I.

Table I. Analysis of Electrolytic Cobalt, Pct

Cobalt	99.55	Sulfur	0.001
Nickel	0.40	Carbon	0.02
Copper	0.005	Lead	0.0001
Iron	0.005		

Sulfide anode sludge treatment

Recovery of the noble metals has always been a principal function of electrolytic refining, and use of sulfide anodes increases the ratio of concentration of these metals over that of metal anode practice.

As stated, the voluminous sludge adheres to the uncorroded remnant of the anode as a cake about 4 in. thick. Separation of sludge from anode scrap is semi-automatic, involving anode bag removal and cleaning, crushing the anode scrap, and separating the sludge from the scrap by wet screening.

As shown in Fig. 3, the anode slime slurry is filtered and washed in two stages on rubber-covered internal drum filters to remove anolyte. The washed sludge, in addition to the precious metals, contains the following percentages of other elements:

Elemental sulfur	97.0	Copper	0.3
Sulfide sulfur	0.7	Selenium	0.15
Nickel	1.25	Moisture content	20.0
Iron	0.6		

Removal of elemental sulfur and selenium by melting and filtering yields what is termed primary anode residue. Melting is continuous in a steam-heated, insulated concrete tank, equipped with a turbine agitator. Melter capacity is about 8000 lb of wet filter cake per hr. The molten slurry is pumped to a steam-heated Kelly filter, from which the clear filtrate is run to holding tanks at 275°F for subsequent sulfur-selenium separation.

Precious metals recovery

Filter cake contains about 50 pct S and all the precious and base metal values of the anode sludge. The material is ideal for remelting and casting into secondary sulfide anodes. To date, electrolysis of such anodes has been applied on a pilot scale, while the bulk of the primary slime is being treated by the conventional process for metal anode residue.

During melting, some sulfur volatilizes and the secondary sulfide anodes contain about 50 pct Ni, 20 pct Cu, 2 pct Fe, 0.5 pct Ag, and 0.3 pct precious metals. These are electrolyzed in a manner and under conditions similar to the primary nickel sulfide anodes, but in a separate circuit.

The anode sludge, produced from the secondary anodes, contains about 90 pct elemental S, which is recovered by filtration as from the primary anode sludge. The residue, containing about 10 pct precious metals, is chemically processed for the recovery of a precious metals concentrate.

Sulfur-selenium separation

The sulfur filtrate from the molten sludge contains about 0.15 pct Se. This element, of the same chemical family as sulfur, is intimately associated with it in the sulfide ore and remains so throughout the various operations.

Blaw-Knox was engaged to engineer and construct a sulfur distillation plant* having a capacity of 100 tpd of pure sulfur.

* Patents pending.

The column is of special design, fabricated of type 309 stainless steel, and equipped with 60 bubble-cap trays. The crude sulfur is pumped at 275°F from the holding tanks to the column. A molten salt heat-transfer medium, externally heated in an oil-fired furnace, supplies heat to the reboiler. Vapors rise from the reboiler at 900° to 940°F, and the column operates at a reflux ratio of 2.5. The sulfur condenser is cooled by circulating heat-transfer salt. Heat removal from this system is through a waste heat boiler operating at 600°F.

Product from the condenser at about 800°F flows continuously to a surge tank where it is quenched in sulfur at 275°F. The contents of the surge tank are cooled by circulation through a waste heat boiler. Part of the circulating stream is continuously drawn off to a rotating shelf-type sulfur cooler. Here sulfur is solidified on water-cooled trays to a cake some 0.75 in. thick. This product, containing less than 5 ppm Se, is broken up, discharged, and elevated to a 100-ton storage bin, from which it can be loaded into railroad cars or trucks. This sulfur has an unusually low ash, bitumens, and acid content.

Selenium is concentrated in the reboiler to about 20 pct, and periodically a fraction is tapped into a cooling tank where the temperature is lowered to about 275°F. It is then drained into drums and shipped to the company's copper refinery at Copper Cliff, Ontario, where it is processed for recovery of pure selenium.

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IRON AND NICKEL BY CARBONYL TREATMENT

A new look at an old problem: How to get iron and nickel out of lateritic deposits economically.

by R. M. Lewis, J. W. Cookston, L. W. Coffer, and F. M. Stephens, Jr.

LATERITIC nickeliferous deposits of Cuba, Celebes, New Caledonia, and the Philippines have attracted the attention of metallurgists for many years. These deposits are quite extensive and contain from 30 to 55 pct Fe, 0.5 to 2.0 pct Ni, 0.5 to 3.0 pct Cr, and up to 0.2 pct Co. The iron occurs principally as the hydrated oxide, limonite, but also occurs as complex carbonates and silicates. The presence of nickel and chromium minerals has been reported by some investigators, but the consensus is that most of the nickel, chromium, and cobalt values occur either in solid solution or are substituted in the iron minerals.

Development of an economic process for the recovery of the values contained in the lateritic ores has been going on for at least 20 years. The absence of discrete nickel, cobalt, or chromium minerals has, of course, precluded the use of strictly physical beneficiation procedures for their recovery. However, numerous attempts have been made to recover nickel as a product or to remove the deleterious nickel and chromium to produce a satisfactory iron ore by physical beneficiation procedures. Pyrometallurgical treatments have also been tried. The Japanese used considerable tonnage of the Philippine ore during World War II to produce a pig iron that was reported as unsatisfactory because of the high nickel and chromium content. Attempts at selective smelting of the laterites to produce a ferronickel alloy apparently have not been economically successful. Numerous patents have been issued for hydrometallurgical processes.

At the present time, the only process used commercially is the ammonia leach process. This process consists of a selective reduction of the ore to produce elemental nickel, which is leached from the ore with ammonium carbonate. Nickel carbonate is precipitated from the leach liquor as a final product. About 75 to 85 pct of the nickel in the ore is recovered in the process. Iron is not recovered, nor is any attempt made to recover the cobalt and chromium except for incidental leaching of the cobalt during the nickel leaching operation. This process is currently in operation at the Nicaro plant in Cuba.

Properties of carbonyls attractive

Among the possible methods that might be used in processing the lateritic ores are the carbonyl processes which utilize the striking physical properties of the metal carbonyls. The metals of Group VI and the transition metals combine with carbon monoxide to form compounds that are characterized by their extreme volatility and their instability with heat. For example, iron carbonyl, $Fe(CO)_5$, melts at −21°C and boils at 103°C; nickel carbonyl, $Ni(CO)_4$, melts at −25°C and boils at 43°C and at temperatures above 150°C both compounds decompose to give the metal and carbon monoxide.

Ludwig Mond was the first to patent a carbonyl process. The patent was issued in 1891 and covers the formation of nickel carbonyl by passing carbon monoxide over reduced nickel ore at a temperature of 0 to 150°C.

A modification of the Mond process was used by the Germans during World War II to produce nickel and iron powders for use in powder metallurgy.

The Mond process has been applied to Cuban lateritic ores as indicated by a U. S. patent issued to K. M. Simpson in 1940. The patent states that the ore was reduced at low temperatures (500° to 700°C) and then cooled to 40 to 80°C and treated with carbon monoxide to form iron and nickel carbonyls. The carbonyls were then decomposed simultaneously at temperatures from 200° to 240°C to form an iron-nickel alloy. Although the process had never reached the commercial stage, it appeared to be worthy of further investigation.

Thermodynamic studies

In order to determine the theoretical conditions for a carbonyl treatment of the laterite ores, a study was made of the thermodynamics of the reactions involving nickel and carbon monoxide and iron and carbon monoxide. Data on the free energy of formation of the carbonyls were incomplete, but experimental values for the heats of formation of iron and nickel carbonyls were available as was a heat content function of nickel carbonyl. The heat content function of iron carbonyl was assumed to be the same as that of nickel carbonyl, and the energy of formation at absolute zero was calculated for each compound. The free energy functions were computed by statistical mechanical equations using the values of the molecular weights and vibrational frequencies of the compounds involved in the following reactions:

$$Ni + 4CO = Ni(CO)_4$$

$$Fe + 5CO = Fe(CO)_5$$

From these data, standard free energies of formation, in calories per mole, were calculated at three different temperatures. These values and the corresponding equilibrium constants are shown in Table I. These are approximate, because of possible inaccuracies in the experimental values for the heat of formation of the carbonyls, and because of the assumption that the heat content of iron carbonyl is the same as that of nickel carbonyl. Because of the decrease in volume of gases during carbonyl formation, it should be expected that the equilibrium yield of the carbonyls would be dependent on pressure. The calculated equilibrium yields are shown in Table II.

These data indicate that nickel carbonyl should be formed more readily than iron carbonyl; however, they are based on equilibrium values and do

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not reflect the kinetics of the reactions. The data also indicate that both reactions are more complete at low temperatures. As the temperatures of the two reactions are increased, the pressure necessary for a relatively high yield of carbonyl product increases, but it is probable that the rate of reaction is increased also. The important consideration is that it is thermodynamically possible to obtain high yields of carbonyl products at relatively low temperatures and pressures, and probably in reasonably short treatments, if the kinetics of the reaction can be accelerated.

Ores for experimental work

Samples of several different ores and concentrates were subjected to the carbonyl treatment. Table III shows the analyses of samples.

Sample 1 is similar to the material now being treated in Cuba at the Nicaro plant; it differs from the rest of the samples in that it is a mixture of a limonitic ore and a serpentine ore in the ratio of 2:1. This sample was used in the preliminary test work.

Sample 3 was used in the later work, including most of the large-scale laboratory work and all of semicontinuous, bench-scale operations. This sample consists almost entirely of limonitic material. The rest of the lateritic ore samples were also limonitic in character and were used to demonstrate the application of the carbonyl process to ores of various compositions.

All of the samples were crushed to pass through a 65-mesh screen before treatment in the process. For some of the work the −65-mesh material was formed into −½ + ¼-in. pellets.

Equipment and procedure

All of the reduction work was accomplished by heating the samples in a hydrogen atmosphere. Crushed samples were reduced in a Vycor pipette, and pelletized samples were contained in a stainless steel basket and reduced in a quartz tube. The samples were heated to the desired temperature in a horizontal tube furnace. The pipette of quartz tube was flushed with nitrogen, and then the hydrogen flow was started and maintained for the desired treatment time. The pipette or quartz tube was then removed from the furnace, and the sample was cooled in a hydrogen atmosphere.

Temperature of the furnace was regulated by a Foxboro controller with a thermocouple between the pipette and furnace wall. Another thermocouple in the center of the charge transmitted the temperature of the sample. Hydrogen flows were measured by a rotameter.

The reduced samples were highly pyrophoric and extreme care had to be taken to prevent oxidation of the samples when they were transferred to the carbon monoxide treatment reactor. A nitrogen-filled dry box was used when assay samples were weighed.

The carbon monoxide treatments were made in a fluid-bed type reactor. The reactor used for treatments at atmospheric pressure was a glass tube 20 in. high, 1½ in. in diam. The tube was expanded to a diameter of 3 in. in the top 3 in. and tapered to a diameter of 0.4 in. in the bottom 4 in. Carbon monoxide entered at the bottom of the tapered section through a tube set at right angles to the section. Gases left the reactor through a tube in the top and passed through a 4-in. glass cyclone and then through a 3-in. filter packed with glass wool. The bottom half of the reactor was submerged in a water bath.

For treatments at pressures above atmospheric, a reactor with the same internal dimensions as the glass reactor was constructed from mild steel tubing with a wall thickness of ½ in. The inner surfaces of the reactor were copper plated, and the piping and fittings were either brass or stainless steel. A removable glass-wool filter in the top of the reactor replaced the cyclone and filter used on the glass reactor, and strip heaters were used instead of the water bath.

The degree of conversion of metals to carbonyls was calculated from the weights and assays of the starting samples and the residues of the treatments.

Investigations with hydrogen reduction

Investigations with ores began with a study of the conditions required for the reduction to metal of the iron and nickel content of Ore 1. It was found that 95 pct of the nickel and 90 pct of the iron could be reduced with 10 times the stoichiometric quantity of hydrogen in a 2-hr treatment at 700°C. Treatments at temperatures up to 900°C did not increase the quantity of reduced metal appreciably, and at

Table I. Thermodynamic Relations

Temperature			F Standard Free-Energy Change, Cal per Mole		Equilibrium Constant, K	
°F	°C	°K	$Ni(CO)_4(g)$	$Fe(CO)_5(g)$	$Ni(CO)_4$	$Fe(CO)_5$
77	26	300	−21,081	−8,369	2.87×10^{15}	1.37×10^{-6}
260.4	126	400	−10,975	+10,085	9.94×10^{5}	3.08×10^{-3}
440.4	226	500	−1,158	+26,057	3.21	4.06×10^{-12}

Note: The calculations are approximate, since the heat formation was estimated to be −50,700 cal per mole $Ni(CO)_4$ from experimental values.

Table II. Equilibrium Yield of $Ni(CO)_4(g)$ and of $Fe(CO)_5(g)$ as a Function of Pressure (1)

Pressure, Atm	Pct by Volume Gas Yields, Temperature 300°K (26°C)		400°K (126°C)		500°K (226°C)	
	$Ni(CO)_4(g)$	$Fe(CO)_5(g)$	$Ni(CO)_4(g)$	$Fe(CO)_5(g)$	$Ni(CO)_4(g)$	$Fe(CO)_5(g)$
1	99.9	94-95	96-97	Zero	40	Zero
5	99.9	98-99	99	0.2	79	Zero
10	99.9	99	99	3	87	Zero
20	99.9	99	99	18	92-93	Zero
50	99.9	99	99	52	96	Zero
100	99.9	99	99.9	70	97-98	Zero
200	99.9	99.9	99.9	83	98-99	0.6
400	99.9	99.9	99.9	90	99	7.0

[1] Fugacity assumed equal to partial pressure.

Table III. Analyses of Lateritic Ores Used in Experimental Program

Sample	Assay, Pct						
	Fe	Ni	Co	Cr	SiO_2	Al_2O_3	MgO
Ore 1	42.1	1.39	0.025	1.96	13.3	—	—
Ore 3	43.4	0.73	0.11	1.70	6.4	10.1	2.3
Ore 5	44.2	0.50	0.02	1.45	16.3	—	—
Ore 8	53.2	1.41	0.19	2.42	4.0	5.5	1.04
Ore 10	41.6	0.72	0.07	1.68	8.4	14.5	1.64
Ore 11	40.0	1.59	0.09	1.99	12.3	16.0	1.51

Table IV. Conversion to Carbonyls, Pct

Treat Time, Hr	Iron		Nickel	
	Dry Process	Wet Process	Dry Process	Wet Process
16	85.5	89.3	93.0	98.1
12	82.7	88.2	94.7	98.5
10	85.1		91.8	
8	85.8	87.2	95.3	97.5
6	83.2	88.0	87.7	97.2
5	82.4	80.9	92.5	92.1
4	77.2	72.0	87.7	89.8
3	72.1		84.4	
2		36.1		49.8

temperatures below 700°C the degree of reduction decreased rapidly. A 4-hr treatment was required to obtain 90 pct reduction of the iron and nickel at 600°C.

Samples of Ore 1 reduced at 700°C were then treated with carbon monoxide. In an 18-hr treatment at atmospheric pressure and at a temperature of 50°C, 18 pct of the nickel and 4 pct of the iron were converted to carbonyls. When the treatments were made under pressure, up to 69 pct of the nickel and 62 pct of the iron were converted to carbonyls with a 6-hr treatment at 100°C and a pressure of 375 psig.

Two adverse conditions were noted in this work. The first was thought to be caused by the auto-reduction-oxidation of carbon monoxide to produce carbon and carbon dioxide followed by an oxidation of the metal. This reaction occurred so rapidly with Ore 1 that sparks were produced when the reduced ore was placed in a stream of carbon monoxide. During these reactions, local temperatures within the charge rose to a point at which particles of the ore coalesced to form spheres. The spheres, in turn, fused together to form slag chimneys, which in some cases extended the full depth of the ore bed.

In the German operations during World War II, small amounts of ammonia were used to inhibit the catalytic decomposition of carbon monoxide during the decomposition of iron and nickel carbonyls. With this in mind, mixtures of ammonia and carbon monoxide were prepared and used in subsequent treatments. The addition of ammonia almost totally eliminated the slagging condition.

The second condition was that the flow of carbon monoxide through the reactor during the treatment was not sufficient to carry off all of the metal carbonyl products. Carbonyls are liquids under the conditions in which they are formed, and it was assumed that some of the ore sample was submerged in carbonyl and unable to react as readily as the rest of the sample. Another disadvantage was that a prolonged period of flushing was required to remove the liquid carbonyls at the end of a test. These difficulties were overcome by reversing the flow of carbon monoxide through the reactor. In order to do this the ore charge was pelletized and contained in a wire-mesh basket which was suspended in the reactor. With these arrangements the carbonyls could be discharged from the bottom of the reactor as liquids, and the only excess carbon monoxide was that required to flush the carbonyls absorbed in the pellets. It was also found that a flow of carbon monoxide through the reactor was not needed, and it was only necessary to supply sufficient carbon monoxide to maintain pressure in the reactor.

In the first series of tests with the modified procedure, conversions of up to 90 pct of the nickel and 80 pct of the iron to their carbonyls were obtained. The conditions for these tests were a 2-hr reduction at 700°C followed by a 16-hr treatment at 100°C under a carbon monoxide plus ammonia pressure of 375 psig. In several of the tests, the liquid carbonyl was discharged at regular intervals. The volumes of carbonyl discharged indicated that 45 pct of the iron and nickel was converted in 1 hr, 60 pct in 2 hr, and 70 pct in 3 hr.

Effect of temperature: The effect of temperature in the reduction of $-\frac{1}{2} + \frac{1}{4}$-in. pellets of Ore 1

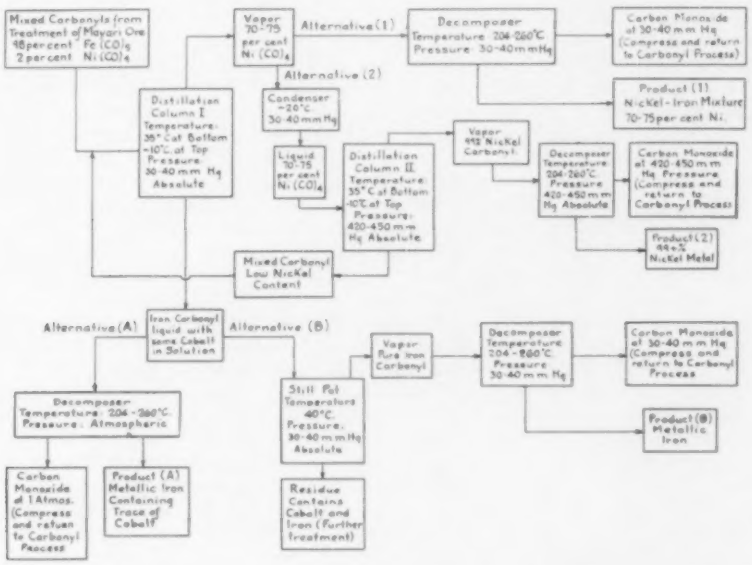
Mixed Carbonyls from Treatment of Mayari Ore 98 per cent Fe(CO)5 2 per cent Ni(CO)4
Vapor 70-75 per cent Ni(CO)4
Alternative (1)
Decomposer Temperature 204-260°C Pressure 30-40 mm Hg
Carbon Monoxide at 30-40 mm Hg (Compress and return to Carbonyl Process)
Alternative (2)
Product (1) Nickel-Iron Mixture 70-75 per cent Ni
Distillation Column I Temperature: 35° C at Bottom -10°C at Top Pressure 30-40 mm Hg Absolute
Condenser -20°C 30-40 mm Hg
Vapor 99% Nickel Carbonyl
Liquid 70-75 per cent Ni(CO)4
Distillation Column II Temperature: 35°C at Bottom -10°C at Top Pressure: 420-450 mm Hg Absolute
Decomposer Temperature 204-260°C Pressure 420-450 mm Hg Absolute
Carbon Monoxide at 420-450 mm Hg Pressure (Compress and return to Carbonyl Process)
Product (2) 99+% Nickel Metal
Mixed Carbonyl Low Nickel Content
Alternative (A)
Iron Carbonyl liquid with some Cobalt in Solution
Alternative (B)
Vapor Pure Iron Carbonyl
Decomposer Temperature 204-260°C Pressure 30-40 mm Hg
Carbon Monoxide at 30-40 mm Hg (Compress and return to Carbonyl Process
Product (B) Metallic Iron
Decomposer Temperature: 204-260°C Pressure: Atmospheric
Still Pot Temperatures 40°C Pressure: 30-40 mm Hg Absolute
Carbon Monoxide at 1 Atmos. (Compress and return to Carbonyl Process
Product (A) Metallic Iron Containing Trace of Cobalt
Residue Contains Cobalt and Iron (Further Treatment)

Fig. 1—Flowsheet for the separation of iron and nickel carbonyls.

was investigated. In this investigation 70 g of the pellets were treated for 2 hr with 10 times the theoretical quantity of hydrogen at temperatures from 650 to 900°C. The reduced pellets were then treated for 16 hr at 100°C under a pressure of 375 psig of carbon monoxide containing 0.3 pct NH_3.

Results show that for a treatment time of 2 hr, superior results are obtained in the carbonyl step when the reduction temperature is within the range of 700° to 850°C. When the reduction is carried out beyond this range, inferior conversions are obtained. The data also show that below 700°C reduction is incomplete and that, at temperatures above 850°C, some further action occurs that has an adverse effect on carbonyl formation. Improved conversions were obtained at lower temperatures by increasing the retention time, or at higher temperatures by decreasing the retention time. However, in no case were the conversions as good as those obtained from the 2-hr treatments at 800°C.

Effect of ore characteristics: The effect of ore characteristics on the reduction and carbon monoxide treatments was investigated with all of the ore samples shown in Table III. Samples of the laterite ores were reduced for 2 hr at 800°C and then were given identical carbon monoxide treatments. All but two of the samples reacted to the treatments in about the same manner. As there is no apparent difference in the mineralogical make-up of the ores, the difference in response of the two ores must be attributed to differences in the physical character of the ores.

Other factors: Other important factors in the reduction step are the quantity of reductant, the cooling conditions, and the time lapse between the reduction and carbon monoxide treatments.

In the light of the preceding work, the standard reduction treatment for laterite ores for subsequent work was a 2-hr treatment with 10 times the theoretical quantity of hydrogen at a temperature of 800°C. The reduced ore was to be cooled in hydrogen and subjected to carbon monoxide treatment immediately.

Effect of ammonia

The first use of ammonia in this work was to inhibit the reaction $2\,CO \rightarrow CO_2 + C$ and prevent the slagging of the ore which occurred when straight carbon monoxide was used. Mixtures of carbon monoxide and ammonia containing from 3 to 10 pct of NH_3 were found to be equally effective for that purpose. Further investigation of the effect of ammonia on the carbonyl-formation reactions indicated that the action of ammonia involves more than just the inhibition of the $2CO \rightarrow C + CO_2$ reaction.

It is our belief that ammonia serves various purposes in the formation of the carbonyls. The reduction-oxidation reactions and slagging of the ore which occur when straight carbon monoxide is used are almost totally prevented by the presence of ammonia. The ammonia may act as a catalyst for the carbonyl formation reactions and also may stabilize the carbonyls after their formation. The mechanism involved is thought to be: 1) Replacement by ammonia of the hydrogen which is adsorbed in appreciable quantities on the surface of the freshly reduced metal; 2) Chemisorption of carbon monoxide by the ammonia until a sufficient concentration of carbon monoxide is available at the reaction zone for carbonyl formation, or 3) Ammonia entering into the reaction to form compounds of the form

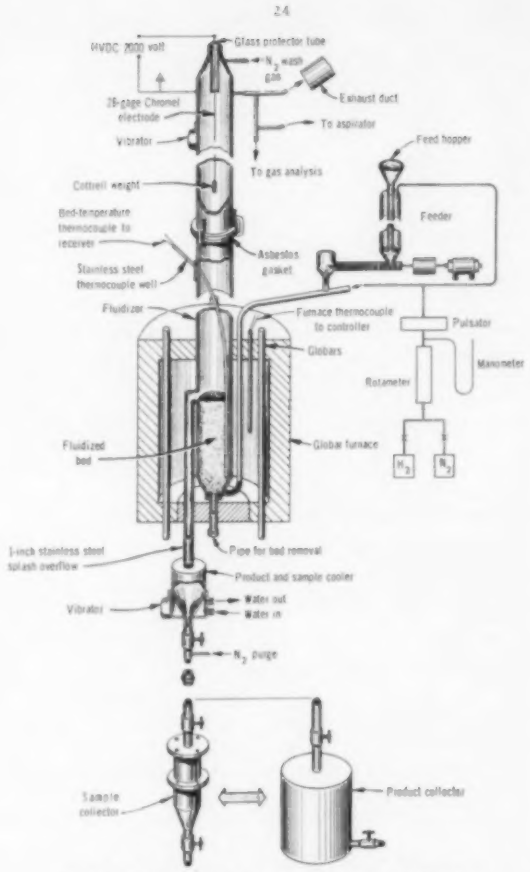

Fig. 2—Fluidized-bed reactor for reduction of Ore 3.

$Fe(NH_3)_x(CO)_{5-x}$ in which the ammonia is successively replaced until the true carbonyl is formed. Whatever the mechanism, the final carbonyl products are free of ammonia, and the ammonia is retained on the solid residues from the treatment.

Formation of additional carbonyls

In the carbon monoxide treatment of a properly reduced ore, temperature and carbon monoxide pressure are the most important considerations. Another factor, noted in the preliminary work and which is expounded upon in the literature, is the adverse effect of carbonyls in the reaction zone upon the formation of additional carbonyls. For this reason, the reactor was provided with a well to which the carbonyls formed in the reactions could be drained and from which they could be removed from the reactor. There was still considerable carbonyl absorbed on the ore, however, which had to be flushed from the ore with carbon monoxide at the conclusion of the carbon monoxide treatment. It was reasoned that this absorbed film of carbonyls must act as a carrier for carbon monoxide, as conversions of up to 85 pct of the iron and 95 pct of the nickel could be obtained in 8 hr or less. If this were true, then a treatment of a slurry of reduced ore in carbonyls should be possible. In order to check this, a reduced sample of −65-mesh ore was slurried with mixed iron and nickel carbonyls. The slurry was stirred in an autoclave under a carbon monoxide pressure of 375 psig and at a temperature of 100°C for 16 hr. At the end of the treatment

the slurry was filtered, and the filter cake was washed free from carbonyls with either ethyl alcohol or carbon monoxide. Conversions of 65 pct of the iron and nickel were obtained. These conversions were the same as those obtained with the dry process under the same conditions. The tests were then duplicated, except that the reduced ore was flushed with ammonia before it was slurried with carbonyl. These tests gave 85 pct conversion of both the iron and nickel which again were comparable to those obtained with the dry process under similar conditions. Results were surprising because of the many statements in the literature about the inhibiting effect of carbonyls on the formation of additional carbonyls.

Temperature, pressure, and time

The slurry treatment was designated the wet process, and both the wet and dry procedures were used in investigating the effects of temperature and pressure on the carbonyl formation reactions.

Effect of temperature: The first investigation of the effect of temperature in the carbon monoxide treatment indicated that 100°C was near the optimum temperature for the formation of iron carbonyl, but that higher temperatures might be more favorable for the formation of nickel carbonyl. In later tests with the dry process, using a carbon monoxide pressure of 375 psig and a treatment time of 16 hr, the same conversions of iron and nickel (Fe, 80 pct; Ni, 90 pct) were obtained at temperatures of 100°, 110°, 120°, and 130°C. At 145°C the conversions were lower by about 25 pct.

With the wet process under a pressure of 375 psig in 18 to 20-hr treatments, the data show that temperature, between 100° and 120°C has no effect on the quantity of iron converted to carbonyl. The equilibrium point of the nickel carbonyl-formation reaction, however, is lowered at temperatures below 110°C which was considered the optimum temperature at 375 psig and was used in all subsequent work.

Effect of pressure: The effect of pressure in the carbon monoxide treatment was studied with the wet process only. Slurries were treated at a temperature of 110°C for 4 hr. The data show that the maximum conversions of iron and nickel are obtained in 4 hr or less at pressures of 450 psig or higher. Previous work has shown that maximum conversions can be obtained at the lower pressures, if the treatment time is prolonged. The inference is that pressure, within the limits investigated, affects the rate at which the reactions approach equilibrium, but does not shift the equilibrium point appreciably.

Standard treatment conditions were taken as a 2-hr reduction at 800°C, an ammonia flush of the reduced ore, and a carbon monoxide treatment at 110°C under a pressure of 375 psig. With these conditions, an investigation of the rate of conversion was made and evaluated by the total conversion obtained in varying periods of treatment with both the dry and wet processes. Results are shown in Table IV.

The data show that practically all of the conversion of the iron and nickel occurs in the first 8 hr of treatment with the dry process and in 6 hr with the wet process. The reactions are somewhat faster with the wet process, as it requires from 75 to 90 min of the total treatment time to bring the autoclave to reaction temperature, while the dry process reaction temperature is attained in 15 min. The data also show an increase of about 4 pct in the conversions with the wet process over those with the dry process.

The data from the temperature and pressure work agree with the data from the thermodynamic studies. All of the data lead to the following conclusions: For any given pressure, there is an optimum temperature at which the carbonyl formations reactions take place. At a given temperature and pressure, there is a maximum conversion which can be obtained, regardless of the time of treatment after the maximum conversion is attained. The reaction time required to obtain a given yield of carbonyl varies inversely with the pressure. Economic considerations are, therefore, the limiting factors on the conditions of temperature and pressure. A pressure of 375 psig and a temperature of 110°C appear to be the minimum conditions under which acceptable conversions of iron and nickel can be obtained in a reasonable reaction time.

Separation of carbonyl mixtures: Cuban laterite No. 3 contains 43.4 pct Fe iron and 0.73 pct Ni. If all the iron and nickel were recovered as carbonyl, the mixture produced should contain 0.38 pct by weight or 1.47 pct by volume of nickel carbonyl. This solution is relatively dilute in nickel carbonyl so that the problem of making a clean separation might have been a difficult one.

The vapor pressure of nickel carbonyl is about seven times as great as that of iron carbonyl at

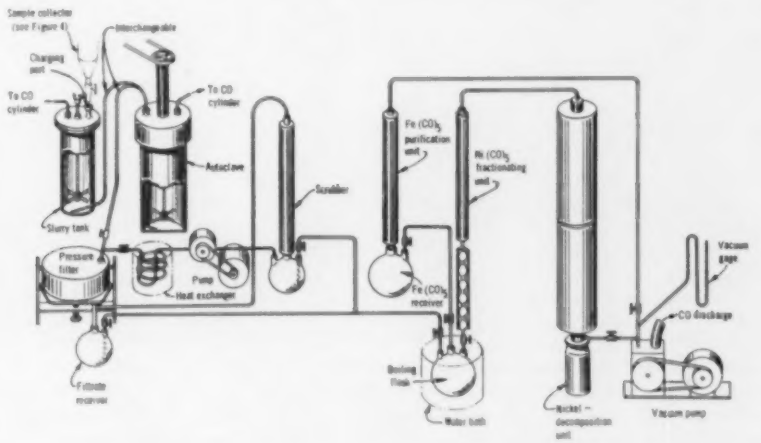

Fig. 3—Apparatus for bench-scale, semicontinuous production of iron and nickel carbonyls.

−7°C and ten times that of iron carbonyl at 43°C, where nickel carbonyl boils at 1 atm. This large difference in vapor pressure suggests that distillation of the mixture and fractionation of the condensate should produce concentration of nickel carbonyl in the first fraction, so that the use of the proper distillation column would permit separation of iron and nickel carbonyls.

Experiments showed that iron and nickel carbonyls obey the gas and ideal solution laws well enough to use theoretical calculations as a first approximation of results to be expected from distillation. However, because of the decomposition of the carbonyls at the temperatures required for distillations at atmospheric pressure, further work was restricted to distillations at lower temperatures and reduced pressures.

Distillation at reduced pressure: Calculations, based on Raoult's Law and the assumption of ideal solution behavior by carbonyl mixtures, indicate that a mixture of 98 pct iron carbonyl and 2 pct nickel carbonyl by weight, should boil at 25°C under a total absolute pressure of 44 mm Hg and at 43°C under an absolute pressure of 91 mm Hg. Calculations indicate that the vapor above a solution containing 2 pct by weight of nickel carbonyl should contain 18 pct by weight of nickel carbonyl when the solution is at 25°C. By the use of a suitable distillation column operating under a partial vacuum, separation of iron and nickel carbonyls from such a solution would appear theoretically possible.

Actual tests in the laboratory using a reflux column made of two condensers and having a bottom temperature of 19°C and a top temperature of −10°C produced vapors, leaving the top of the column, containing 90 to 95 pct nickel carbonyl. Because of the small amount of nickel carbonyl used, no attempt was made to condense this to a liquid, but it was decomposed directly by passing it through a hot tube to produce nickel-iron alloy assaying as high as 98.5 pct Ni. With these conditions there was no decomposition of either carbonyl in the distillation equipment, and a nickel-free iron carbonyl containing trace amounts of cobalt was produced. A pure iron carbonyl can be prepared by distilling over the iron carbonyl at 40°C and 30 to 40 mm Hg. The cobalt remains in the still-pot as a solid iron-cobalt compound of variable composition. Fig. 1 shows a flowsheet for the iron, nickel carbonyl separation procedures.

Decomposition

Work on the decomposition of iron and nickel carbonyls was preliminary in nature and was not concerned with the development of commercial procedures; primary concern was to determine the temperature and pressure conditions under which decomposition of the carbonyls could be obtained.

Most of the decomposition work was done in glass, aluminum, or iron tubes. The tubes were heated externally by electrical tube furnaces. The carbonyls were introduced to the decomposition tubes as vapors or sprays and with or without carbon monoxide as a carrying medium. Exhaust gases from the tubes were passed through cyclones, filters, and condensers to collect the solid decomposition products and any undecomposed carbonyls.

Nickel carbonyl was decomposed at temperatures from 200° to 260°C and under pressures from 30 to 760 mm Hg. No evidence of decomposition of nickel carbonyl in the gas phase was observed. All deposits were metallic films. At the lower temperatures and pressures there was some codeposition of carbon.

Iron carbonyl was decomposed at temperatures from 250° to 300°C and under pressure from 30 mm Hg. to 150 psig. Iron carbonyl tended to decompose in the gas phase at the higher temperatures. However, if sufficient surface was available the iron would deposit as an adherent film. Carbon content of the iron was usually less than 1 pct.

Bench-scale, semicontinuous process

All of the steps leading to the production of nickel and iron metals by a wet carbonyl process had been shown to be technically feasible in small-scale batch operations. Because of the high volatility and instability of the iron and nickel carbonyls, metal balances were not obtained in the batch operations. Some of the batch operations, such as reduction and filter-cake scrubbing, had not been carried out by means suitable for a commercial operation. It was believed that, if the batch operations, from the reduction step through the decomposition of nickel carbonyl, could be integrated in a closed system with equipment adaptable to commercial operations, metal balance would be achieved and information for the planning of a pilot plant obtained. Equipment to achieve these objectives was designed and constructed.

Equipment

The following operations needed to be integrated in a system that would permit the transfer of all solids, liquids, and gases through the system without exposing them to the atmosphere: 1) reduction of ore, 2) cooling of reduced ore, 3) ammonia treatment of reduced ore, 4) mixing of reduced ore with iron carbonyl to form a slurry, 5) carbon monoxide treatment of slurry under pressure, 6) filtration of treated slurry, 7) removal of carbonyls from filter cake, 8) separation of nickel carbonyl from iron carbonyl, 9) decomposition of nickel carbonyl, and 10) purification of iron carbonyl.

Fig. 2 is a line drawing showing the fluidized-bed reactor and auxiliary equipment for carrying out the first three operations. Fig. 3 is a line drawing showing the equipment for conducting the remaining operations. The size of the equipment was based on the capacity of a 1-gal autoclave that was available for this work. The capacity of the system, based on an autoclaving treatment of a slurry containing 33 pct solids in a 10-hr cycle, was 100 g of raw ore per hr.

Operation

Several test runs were made with the equipment for the bench-scale semicontinuous production of nickel metal and iron carbonyl. The reduction equipment was in operation for 230 hr, and 140 kg of ore were reduced. The rest of the equipment was operated for 170 hr, during which time 12 kg of reduced ore were treated.

The unit operations involved in the integrated, bench-scale equipment succeeded in producing conversions and yields comparable to the best results obtained in the batch operations. The successful operation of the integrated unit proved that the individual steps of the process could be combined to give a process that is technically feasible and capable of giving high recoveries of iron and nickel from the laterites.

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AIME

METALLURGICAL SOCIETY OF AIME

NEWS

National Open Hearth Blast Furnace, Coke Oven, and Raw Materials Conferences

A full house was the order of the day at the Annual Fellowship Dinner of the Blast Furnace, Coke Oven, and Raw Materials and National Open Hearth Steel Conferences . . . an attendance of 1592, in spite of the current low operating rates in industry.

Above: Walter R. Hibbard, Jr., President of The Metallurgical Society of AIME greeted guests at the Fellowship Dinner, Tuesday, April 15, 1958. Below, left: M. F. Yarotsky, Chairman, National Open Hearth Steel Committee, presented the F. B. McKune Memorial Award to L. J. Trilli (left) and R. W. Joseph (right) of Inland Steel Co.

Above, left: M. F. Yarotsky, NOHC Chairman, presented the Open Hearth Award to R. E. Stoll, (left) and E. C. Rudolphy (right) of U. S. Steel Corp., Chicago. Seated, left to right, were: Toastmaster, Van H. Leichliter, U. S. Steel Corp., and the guest speaker, Charles M. White. Below, right: F. M. Hamilton, Chairman, Blast Furnace, Coke Oven, and Raw Materials Committee, presented the Journal of Metals Award to H. U. Ross (left), University of Toronto, Canada.

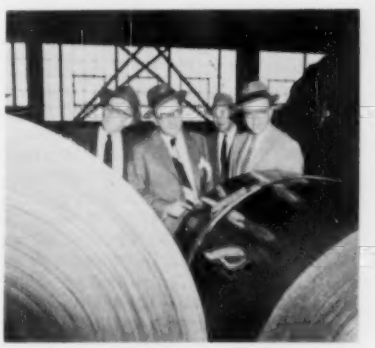
Above, left: AIME Secretary, Ernest Kirkendall, received a scroll in recognition of his service to the National Open Hearth Steel Committee, from M. F. Yarotsky, NOHC Chairman. Seated at left, was Augustus B. Kinzel, AIME President. Below: The featured speaker, Charles M. White, Chairman of the Board, Republic Steel Corp.

(Left) At the speakers table are: (L to R) R. L. Stephenson, Secretary, Blast Furnace, Coke Oven & Raw Materials Committee; E. R. Dean, Chairman, Finance Committee; Blast Furnace, Coke Oven & Raw Materials Committee; J. B. Wagstaff, Chairman, Program Committee, Blast Furnace, Coke Oven & Raw Materials Committee; K. F. Fetters Chairman, ISD; M. F. Yarotsky, Chairman, National Open Hearth Steel Committee; F. M. Hamilton, Chairman, Blast Furnace, Coke Oven & Raw Materials Committee; J. W. Duncan, Vice-Chairman, Blast Furnace, Coke Oven & Raw Materials Committee; C. T. Marshall, Past-Chairman, Blast Furnace, Coke Oven & Raw Materials Committee; Ernest Kirkendall, Secretary, AIME; R. C. Mohr, Chairman, Membership Committee, Blast Furnace, Coke Oven & Raw Materials Committee; the late C. L. Corban, Vice-Chairman, Blast Furnace, Coke Oven & Raw Materials Committee; and R. W. Shearman, Secretary-Treasurer, The Metallurgical Society of AIME.

Above: Guests visited Republic Steel Corp.'s Cleveland District blast furnace plant, which included Blast Furnace No. 6, one of the largest in the world. At capacity, it produces 1625 tons of iron daily.

A view usually reserved for a melting piece of open hearth scrap.

What a place for an umpire (Inside the new carbon bottom open hearth.)



Above: These coils are great!

Below: That was quite a strip mill.

IF the 1958 AIME Annual Meeting defied New York's worst blizzard of the season, then certainly the National Open Hearth Steel and the Blast Furnace, Coke Oven, & Raw Materials Conference defied the generally depressed economic conditions of the steel industry and came out on top. Shattering all expectations of attendance was the total registration of *1592*.

The meeting was held from the 14th to the 16th of April in Cleveland's Statler Hotel. All arrangements were perfectly laid on, and the meeting proceeded throughout on schedule. Bouquets are certainly due to the local section under C. W. Cravens' chairmanship, and additional ribbons are due to the top-notch program committees of both conferences for arranging what was one of the most interesting and stimulating schedules of technical sessions yet! One registrant from England was heard to comment "If you people keep up with such fine programs as these, I simply can't afford to stay away, even traveling this distance."

Blast Furnace Sessions

Technical Sessions of the Blast Furnace, Coke Oven, & Raw Materials conference began with two particularly stimulating papers: one on operation with humidified blast by Wilson of Inland Steel (April JOURNAL OF METALS) and the other describing Jones & Laughlin's work with steam and oxygen in the blast furnace.

Such recent developments as atommatic stove changing and desulfurization of hot metal were the subject of two papers, while P. E. Cavanagh of the Ontario Research Foundation presented a paper on Canadian iron ore.

At the first Coal Preparation session, a French engineer, E. M. Burstlein described his process for coke production from bituminous and non-bituminous coals. Other papers were on constituents of a coke blend and carbonizing tests. And at the final joint Blast Furnace and Coke Oven session a paper on the German Walsum Iron-Coke Process (April JOURNAL OF METALS) was given and followed by quite a lively discussion on the whole problem of coking lower grade coals. This was followed by a panel discussion: *Effect of Coke Size on Blast Furnace Production.*

The Annual Luncheon and Business Meeting of the Blast Furnace, Coke Oven & Raw Materials Committee was held on Tuesday, April 15, with chairman F. M. Hamilton as Toastmaster. Treasurer Robert W. Shearman reported that the committee had reached the *black* with the fastest growing surplus in the Institute, some $6000 at present. This has been the result of increased advertising for the Conference Proceedings as well as higher registration fees. From the membership chairman there was a report of 57 new members signed up at the meeting. Nominated and approved for 1958 officers for the committee were: James W. Duncan, Chairman; Clarence L. Corban, Vice Chairman; Wilbert Huntley, Secretary; Robert W. Shearman, Treasurer; John Poast, four-year director; A. D. Shattuck, four-year director; and J. B. Wagstaff, two-year director.

Open Hearth Sessions

Many interesting papers were presented at the Open Hearth technical sessions, including those on the use of elevators and mechanized scrap charging at Armco and the streamlining of charging cycles at Gary works of U. S. Steel Corp. A panel discussion on the results of using

(Continued on page 429)

New England Regional Conference Held

The Twelfth Annual AIME New England Regional Conference was ushered in at the General Electric Research Laboratories, Schenectady, N. Y., on April 18 and 19, 1958, with a registration of 150. *Thermal Processing of Metals* was the theme of the Conference which was sponsored by the Boston and Connecticut Sections of AIME, in cooperation with members from eastern New York State.

The lead-off paper on the Metals Science Session, April 18, was *Thermal Motion of Defects in Metals*, presented by C. E. Birchenall of Princeton University. K. T. Aust, General Electric Co., followed with a paper on *Thermally Activated Motion of Interfaces. Recrystallization* was then the subject of a talk by C. G. Dunn, General Electric Co. J. L. Walter, General Electric Co., gave the final paper of the session, *Preferred Orientation.*

Technical chairman for the session was F. S. Chudy of the Bullard Co.

The Metal Engineering Session commenced on April 19, with papers dealing with the engineering aspects of thermal processing of metals. W. A. Anderson, Aluminum Research Laboratory, Aluminium Co. of America, presented the initial paper, *Effect of Orientation Textures on Mechanical Properties.* H. C. Fiedler, General Electric Co., ensued with a paper on *Effect of Orientation Texture in Magnetic Properties.* Subsequently, J. E. Burke, General Electric Co., delivered a paper, *Structure Control of Ceramics Systems.* The terminating paper, *New Developments in Furnace Atmospheres*, was given by G. O. Underwood and R. B. Gilliland of the New England Metallurgical Corp.

The session was chairmaned by L. R. Frazier of Metals Hydrides Inc.

A highlight of the Conference was an exhibition of recent research at General Electric in the fields of metallurgy, ceramics, and solid state chemistry.

A keynote of the Conference was an after lunch visit on April 18, to the Metal and Ceramics Building of the General Electric Research Laboratory. Completed in 1955, the laboratory was described in an orientation talk by Robert McKechnie, as the mechanism for transition between basic research and profit-making activities. The objectives are to develop new and improved materials and processes, to perfect some of them as new business, and to prepare experimental samples of special metals and ceramics.

General Electric Co.'s large steam-turbine generator department was also visited. Dolf G. Ebling, in an orientation talk explained that the plant covers 22 acres under one roof, and produces a very few items at a high unit price, namely about 50 to 60 turbine-generators per year. However, the average monthly output will supply the electric power and light requirements for a city of 1,000,000 population.

The Van Curler Hotel, Schenectady, was the scene of the Fellowship Dinner, held the evening of April 18. Guest of honor was Augustus B. Kinzel, President of AIME. J. Herbert Holloman, manager of metallurgy and ceramics research dept., General Electric Co., served as Toastmaster. The featured speaker was Vincent J. Schaefer, director of research for Munitlap Foundation. In conjunction with his talk *Adventures in Meteorology*, Schaefer presented motion pictures in color to show startling meterological effects. The movie presentation included the trigger effect of cloud seeding with dry ice or silicon iodide, a development that has made Dr. Schaefer famous throughout the world.

New officers were elected at a business meeting held on the afternoon of April 19. D. L. Martin, General Electric Co., was elected chairman for the 1959 New England Regional Conference. Other officers elected were: J. P. Lynch, Jr., American Brass Co., vice-chairman; C. M. Adams, Massachusetts Institute of Technology, secretary; P. J. Clemm, General Electric Co., treasurer; and R. M. Treco, and W. P. Mathew, members-at-large.

D. E. Trout, Scovill Mfg. Co., retired as Conference chairman. Assisting him in technical program planning were D. L. Martin, R. M. Treco, and K. T. Aust, and in plant trip and dinner arrangements, P. J. Clemm.

Hot Laboratories and Equipment Conference

Abstracts for papers to be presented at the Seventh Conference on Hot Laboratories and Equipment to be held in Cleveland, April, 1959, are due September 15, 1958. They should be sent to program chairman: *L. G. Stang, Jr., Brookhaven National Laboratory, Upton, New York.* Deadlines for the papers will be announced later.

Papers are invited on all phases of hot laboratories and equipment for handling radioactive material, including design, construction, operation, maintenance, decontamination, remodelling, shielding calculations costs, etc.

Physical Chemistry of Process Metallurgy Symposium Scheduled

An International Symposium on the Physical Chemistry of Process Metallurgy was recently announced by John Chipman, Massachusetts Institute of Technology, the President Elect of The Metallurgical Society of AIME. The symposium, scheduled for April 27-30, 1959, in Pittsburgh, is being sponsored by The Metallurgical Society of AIME in cooperation with the Physical Chemistry of Steelmaking Committee of the Iron and Steel Division, the Physical Chemistry of Extractive Metallurgy Committee of the Extractive Metallurgy Division, and the Melting and Casting Committee of the Institute of Metals Division.

JOHN CHIPMAN

The four-day symposium will cover the following general topics: **I) Physical Chemical Principles**—a) theory and behavior of metal solutions and of oxide phases; b) electrochemistry behavior of fused salts and electrochemical reactions; c) pyrochemistry reactions between slag, metal, gaseous, and solid oxide phases. **II) Transport Phenomena**—a) principles of kinetics; reaction rates, and mass and energy transport; b) electrochemical phenomena, electrode reactions, and electrolysis; c) pyrochemical reaction mechanisms. **III) Application of Physical and Chemical Principles to Metallurgical Processes Thermodynamics, Kinetics, and Mass and Energy Transfer.**—a) communition; b) leaching, solution and phase separation; c) roasting and agglomeration; d) reduction and smelting; e) gaseous, liquid, and solid phase refining; f) casting and solidification. **IV) Application of Operations**—a) research and statistical methods to process control.

Technical papers in all three areas of process metallurgy are invited for presentation at this Symposium. In addition to having reports on the results of new basic research, it is planned to emphasize in the program the use of fundamental facts

for obtaining better understanding and better control of the new and the traditional methods for producing the various metals.

The deadline for submission of papers is Jan. 1, 1959. Anyone planning to participate in the Symposium should write to *Professor John F. Elliott, Room 8-109, Massachusetts Institute of Technology, Cambridge 39, Mass.*, who is serving as Programs Chairman. Professor C. Law McCabe, Carnegie Institute of Technology, is Conference Chairman.

A proceedings of the Symposium will be issued.

Blast Furnace and Open Hearth Conferences

(Continued from page 427)

Exothermic Ferro Manganese included metallurgists from four firms and resulted in an animated discussion on a very timely topic.

Leo Reinartz, one of the founders of the Open Hearth Conference, presented a paper on *The First Hundred Years of Basic Open Hearth Steelmaking*. Papers on *continuous waste gas analysis*, and *continuous oxygen analysis* were of high interest for those faced with control problems in the open hearth. Further reports on the all-basic open hearth, as well as open hearth expansion problems, were featured in the Refractories and Masonry sessions. A report on the carbon sub-hearth of Republic Steel's Warren, Ohio, plant was particularly interesting, since conferees were later to see, first-hand, a similar hearth in Cleveland.

In the Physical Chemistry of Steelmaking session papers on some of the fundamentals of oxygen in steelmaking were presented by Professor Philbrook of Carnegie Institute and Luerson of Inland Steel. A new idea in oxygen treatment of hot metal, the preheating tower, was described by engineers from Union Carbide.

Fellowship Dinner

The Conference's Annual Fellowship Dinner was held on Tuesday, the 15th of April, in the Grand Ballroom of the Statler Hotel, Cleveland. Following dinner, a roving band of music makers struck up *Volga Boatman*, which was sung in Russian (we guess) by National Open Hearth Steel Chairman, M. F. Yarotsky. . . which got the festivities of the evening off to a good start. Yarotsky presented the *F. B. McKune Memorial Award* to R. W. Joseph and L. J. Trilli of Inland Steel Co. for their paper, *A Study of Wall Erosion in Wide Slab Molds*. *The Open Hearth Conference Award* was won by R. E. Stoll, and E. C. Rudolphy of U. S. Steel Corp.'s South Works for their paper, *Effect of Nozzle Characteristics on Steel Pouring Streams* AIME Secretary, Ernest O. Kirkendall, was presented a scroll for his work on behalf of the National Open Hearth Committee, by Chairman Yarotsky.

Professor H. U. Ross, of the Department of Metallurgical Engineering, University of Toronto, was the recipient of the *Journal of Metals Award* of the Blast Furnace, Coke Oven & Raw Materials Committee. His award winning paper, *Smelting of Titaniferous Ores*, is published in this issue on p. 407. Committee Chairman, F. M. Hamilton, presented to C. T. Marshall, past-chairman, a scroll of appreciation, and another scroll was presented to AIME Secretary Ernest Kirkendall for his work on behalf of this committee.

Festivities were climaxed when toastmaster, Van H. Leichliter, President, American Steel & Wire Div., U. S. Steel Corp., introduced speaker Charles M. White, Chairman of the Board of Directors, Republic Steel Corp. for his talk, *The Great Delusions*.

Plant Trip

The third day of the meeting included a trip through the Cleveland plant of Republic Steel Corp., Ohio's largest steel plant. Blast Furnacemen were given a tour of the blast furnace and coke oven departments, including the number six furnace, one of the largest in the world, capable of producing 1625 tons of iron daily. Open Hearth participants, allowed to spend several hours in the open-hearth shop, were particularly interested in inspecting the new 375 ton open-hearth furnace with its carbon sub-hearth. It was the first time that many persons had been given the opportunity of strolling inside of the furnace itself. Also viewed, was the 98-in. continuous hot strip mill.

NORWEGIAN METALLURGICAL EXHIBITION

Oslo, Norway, will be the site of the *Chemical Engineering-Metallurgy 58* exhibition, to be held on Oct. 16 to 26, 1958. Aimed to spread industrial industrial information in these fields, this will be the first time a large scale exhibition covering chemical engineering and metallurgy has been held in Scandinavia.

Among the offerings to be shown are: apparatus, equipment, and technical aids for use in the laboratory; raw materials and auxiliaries for the metallurgical industries; furnaces, machinery, equipment, and auxiliary equipment for production and production control in a) the smelting industries, and b) industrial processing and treatment; equipment for extraction and enriching of ores; dust and gas extractors; and testing and control of raw materials and products.

In connection with the exhibition there will be a number of study groups dealing with: air and gas purification, and crushing, grinding, and shifting.

The 10 day exhibition is sponsored by *Studieselskapet for Norsk Industri, Forskningsveien 1, Blindern, Oslo, Norway*, in collaboration with the Norwegian Export Council.

PERSONALS

Carpenter Steel Co., Reading, Pa., recently appointed **Samuel E. Tyson** as metallurgist, Stainless Steels. He was previously assistant metallurgist, Stainless Steels.

Edward P. Sandbach, formerly with United Engineering and Foundry Co., was recently appointed chief metallurgist for the Mckintosh-Hemphill Div. of E. W. Bliss Co. In his new position, he will head the metallurgical departments in the division's Pittsburgh and Midland, Pa. plants.

E. L. MacWhorter, Philadelphia Division Manager for A. M. Byers Co., Pittsburgh, retired January 1st. MacWhorter, a leading authority on wrought iron, had served the Byers Co. for 33 years. He is succeeded by **Jed J. Merrill,** who has been New England Division Manager.

REMINDER

The deadline date for submission of abstracts for the IMD sessions at the Fall Meeting of The Metallurgical Society, October 27-30, Carter Hotel, Cleveland, Ohio, is **July 15, 1958.** The abstracts should be between 200 to 300 words, and should be submitted in three (3) copies to:

IMD Programs Committee
The Metallurgical Society of AIME
29 West 39th Street
New York 18, N. Y.

The Iron and Steel Division sessions and the Extractive Metallurgy Division sessions will consist of invited papers.

PRIZE PAPER CONTEST

The Resistance Welder Manufacturer's Association is now offering $2000 in prizes for papers on Resistance Welding, Projection Welding and General Resistance Welding Classifications. The contest which closes Sept. 15, 1958, is open to anyone, without restriction, from the United States, its possessions, and Canada. It is open to students, and instructors, as well as people in all industries.

Papers will be entered in two classifications: Those dealing with the latest developments in *projection welding* will be entered in a special classification; and papers dealing with developments *in all other* types of resistance welding (spot, seam and flash-butt, resistance brazing, etc.) will be entered in the general classification.

The prizes for the Special Projection Welding Classification are: 1st prize—$600, 2nd prize—$300, and 3rd prize—$100. The prizes for the General Resistance Welding Classification are: 1st prize—$600, 2nd prize—$300, and 3rd prize—$100.

Copies of the contests rules and application may be obtained from the *Resistance Welder Manufacturer's Association, 1900 Arch Street, Philadelphia 3, Pa.*

Personals

(Continued from page 429)

Harry E. Thompson was recently promoted to New England Division Manager for A. M. Byers Co., Pittsburgh. He previously was a field service engineer with the Byers' New England Division.

Frank J. Newman, formerly sales manager for the Process Instruments Div., Beckman Instruments Inc., Fullerton, Calif., was recently promoted to marketing manager for the division.

Two key appointments were recently announced in the Industrial Division of Kaiser Aluminum & Chemical Corp., Oakland, Calif. **Arthur H. Branstad** has been named manager of sheet and plate, and **William J. Edmunds, Jr.** has been appointed manager of extrusions and forgings. Both Branstad and Edmunds have been with Kaiser Aluminum for more than ten years.

Edward J. Hanley, president of Allegheny Ludlum Steel Corp., was recently elected to the board of directors of Westinghouse Air Brake Co., Pittsburgh. He is presently serving on the board of directors of several companies and civic organizations.

John H. Krey was recently elected a vice president of United States Foil Co., Richmond, Va., the parent company of Reynolds Metals Co. He formerly was assistant to the president of the company.

Lester E. Russell, previously assistant superintendent of reduction, Reynolds Metals Co.'s reduction plant, Troutdale, Ore., was recently named superintendent of reduction at Reynolds new aluminum plant being built at Rooseveltown, N. Y.

Loring S. Brock has been appointed manager of structural and plate products for United States Steel Corp., Pittsburgh. He formerly was assistant manager of structural and plate products sales for the company.

Louis J. Quitoni, formerly plant superintendent at Bestway Products Inc., Mountainside, N. J., was recently appointed to the New York field Sales Staff of the Press Division of F. J. Stokes Corp., Philadelphia.

W. B. Farnsworth was recently promoted to assistant manager of research, for the Linde Tonawanda Research Laboratory, of the Linde Co., Tonawanda, N. Y. He was previously manager of gas process development at the Linde Newark Laboratories.

Leonard V. Klaybor has been appointed as product manager, tool steel sales, Allegheny Ludlum Steel Corp., Pittsburgh. He was formerly chief research metallurgist, tool and die steels, at the Dunkirk, N. Y. works of Allegheny Ludlum. **Paul R. Borneman** has succeeded Klaybor as chief metallurgist, tool and die steels. He had been a supervising research metallurgist at the Dunkirk Works.

John L. Cross, assistant sales manager of apparatus at Westinghouse Electric Corp.'s headquarters in Pittsburgh, was recently appointed industrial sales manager for the corporation's atlantic apparatus sales region in Philadelphia.

J. Ranald Fox, formerly works manager of the alumina plant at Aluminum Co. of America's Point Comfort Operations, has been named assistant general manager of Alcoa's refining division. **A. B. Kaltwasser,** previously alumina plant production manager at Point Comfort, has succeeded Fox as works manager.

(Continued on page 431)

1957 Transactions Volume Now Available

The 1957 METALS TRANSACTIONS VOLUME is now available. It is priced at $9.00 to non-members of AIME, and $6.30 to members. Other available METALS TRANSACTIONS VOLUMES are 1956, 1955, and 1954. These volumes may be ordered through the AIME Order Department.

One-Week Vacuum Metallurgy Program

Designed for practicing engineers, a one-week summer program on vacuum metallurgy will be held at New York University's College of Engineering, University Heights campus, Bronx, June 23 through 27. It will include 20 lectures and a panel discussion.

Augustus B. Kinzel, vice president for research, Union Carbide Corp., and AIME president, will deliver the keynote address at 9:45 am, Monday, June 23. Also speaking at the first session will be: Harold K. Work, director of the college's research division; John P. Nielsen, chairman of the department of metallurgical engineering; and Rointan F. Bunshah, chairman of the program and associate research scientist at the College.

Among the topics to be covered are: vacuum systems and physical chemistry; special processes—designs and operation, degassing, coating, and quality control, and applications of vacuum processed materials.

There will be adequate opportunity provided for discussion of lecture material. In addition, informal evening discussions are planned during the week.

ROCKY MOUNTAIN MINERALS CONFERENCE

The Newhouse Hotel, Salt Lake City, will be the scene of the 5th Annual Rocky Mountain Minerals Conference scheduled for Sept. 17, 18, and 19, 1958. Heading the list of speakers for the three day meet are Governor George D. Clyde of Utah and Howard C. Pyle, president of the Monterey Oil Co.

Special sessions at the conference will be devoted to minerals benefactions, extractive metallurgy, iron and steel, mining and geology, industrial minerals, and petroleum and coal.

The conference will end Friday night, September 19th with a dinner-dance.

Education

(Continued from page 382)

2000°F. Some of the creep tests will run as long as 1000 hr.

Research on the high temperature properties of ceramic materials will also be undertaken in the new laboratory.

SCHOLARSHIP AID

Lehigh University has received $6,674 through the final distribution of a bequest of the late Mrs. Mary Haydon Hansen of Hazelton, Pa. This is to be added to funds previously left to the University by Mrs. Hansen, uping the total of the endowment fund to $378,436 for scholarship aid.

The endowment, named the James Clark Haydon Scholarship Fund by the university's trustees, was designated by Mrs. Hansen as a memorial to her father, the late James Clark Haydon of Hazelton, Pa., who was a close associate of several Lehigh University alumni and trustees.

A grant of $2500 from the Carpenter Steel Co. was made to Lehigh for the establishment of a graduate fellowship in science and engineering. The grant is to be known as the J. Heber Parker fellowship in memory of the former president of Carpenter Steel Co., Reading, Pa.

$1500 of the total grant will be for the graduate student selected for advanced study and $1000 will go to the university for tuition and expenses supporting this graduate work.

The first award is to be made during the 1958-59 academic year.

Personals

(Continued from page 429)

L. J. Venne has joined the sales department of Electro Metallurgical Co. as a metallurgical service representative. He has been assigned to the foundry section of the metallurgical service division in Cleveland. Venne previously was associated with the United States Steel Corp. and American Steel Foundries.

Kenneth Soper, formerly superintendent of mill products at the Olin Mathieson Chemical Corp.'s New Haven, Conn. plant, is now serving as superintendent of mill products at the East Alton, Ill. plant of Western Brass Mills division of the company.

Meryl B. Draper was recently appointed as general superintendent of the Wallingford, Conn. steel service plant of Joseph T. Ryerson & Son Inc., Chicago. He previously had served the company as general foreman in the firm's Boston plant.

Members of the Acid Converter and Basic Oxygen Steel Committee, Iron and Steel Division, The Metallurgical Society of AIME, are shown grouped before an LD Converter during a plant visit to the Aliquippa works of Jones & Laughlin Steel Corp. on Apr. 30, 1958.

Acid Converter and Basic Oxygen Steel Committee, Guests of Jones & Laughlin

The Acid Converter and Basic Oxygen Committee of the Iron and Steel Division were guests of the Aliquippa works of Jones & Laughlin Steel Corp. on April 30. J. N. Albaugh, superintendent, and C. O. Byers, industrial engineer, greeted the 45 visitors.

At the plant, the Committee witnessed 81-ton heats made with a 35-min refining period using top-blown oxygen. Tap-to-tap time was 45 to 50 min with approximately 20 heats being made per 24 hr. Home scrap constitutes 28 pct of the metallic charge, the balance being hot pig iron. Burned lime and mill scale are also charged. Oxygen is of 99.5 pct purity, cost is blown at a rate of 3500 to 3800 cu ft per min. Effluent gases are cleaned in an electrostatic precipitator. The steel produced is used for tin-plate, continuous butt-welded pipe, rod, and wire products. The oxygen furnaces have been in service since Nov. 11, 1957.

After lunch at the Duquesne Club, Pittsburgh, an off-the-record technical session was held under the leadership of Arthur B. Wilder, chief metallurgist of National Tube Division, United States Steel Corp. In a discussion of the basic oxygen steel process, these topics were covered: iron quality and scrap, oxygen supply, vessel linings, vessel and plant design, blowing practices, deoxidation and recarburization, and fume control. The discussion on the acid bessemer steel process included: hot metal mixers and ladles, blowing equipment, vessel linings, vessel design, and blowing practices.

The Union Carbide Corp. recently appointed three vice presidents in the Bakelite Co. **C. M. Blair** was made vice president, planning; **R. D. Glenn,** vice president, development; and **L. K. Merrill,** vice president, technical, with responsibility for patents and licenses.

William C. Miller, formerly a sales engineer with The Beryllium Corp. of Reading, Pa., is now sales manager of National Precision Casting Corp., Reading, Pa.

High Voltage Engineering Corp., Burlington, Mass., has recently appointed **Richard Chase** purchasing agent, and **Charles Doucette** head of the company's tube and vacuum department.

Ottilie Amminger, formerly in charge of the chemical laboratory at the metallurgical department of L. von Roll, Eisenwerke A.G. in Switzerland, is now associated with Leschen Wire Rope Division, H. K. Porter Co. Inc., St. Louis, as chief metallurgist. She will be responsible for quality control and the new metallurgical laboratory in the Leschen wire-drawing mill.

Davenport West, Jr., with Union Carbide Corp. since 1942, was recently named vice president of the Union Carbide Ore Co. Division.

Irving R. Zwick, formerly assistant treasurer and comptroller of Solar Steel Corp., Cleveland, has been elected treasurer of the corporation.

(Continued on page 432)

Personals

(Continued from page 431)

R. W. Minnig, technical superintendent of Great Lakes Carbon Corp.'s Electrode Division plant at Niagara Falls, N. Y. was recently appointed technical manager of the Anglo-Great Lakes Corp. Ltd., graphite plant, Newcastle-on-Tyne, England. He is succeeded by **R. F. Markel,** former assistant to the chief engineer at Niagara Falls. Other Great Lakes Carbon electrode division promotions of technical personnel are: **A. A. Cline,** previously process engineer at the Morgantown, N. C. plant to technical control manager, Niagara Falls; **C. A. Switzer, Jr.,** from plant engineer Niagara Falls to chief engineer of the division; and **J. P. MacKay,** former assistant plant engineer at Morgantown to plant engineer, Niagara Falls.

H. A. Steinherz was promoted in the fall to the position of manager of engineering and development of NCR Equipment Corp., Newton Highlands, Mass. He was previously a development engineer for the company.

Raymond J. Wean and **Loyd E. Yoder** were recently named to the executive committee of the Board of Trustees of Carnegie Institute of Technology, Pittsburgh. Yoder is vice president of the National Broadcasting Co. and general manager of WRCV and WRCV-TV in Philadelphia. Wean is president of the Wean Engineering Co., Pittsburgh.

Phelps H. Adams was recently appointed as vice president, public relations, for United States Steel Corp., New York City. Adams had been executive director, public relations and assistant to the chairman of the board since 1956.

Lester B. Coleman has been appointed advertising manager for the Flexible Steel Lacing Co., Chicago. In addition to his new assignment, Coleman will continue to handle various sales department duties.

Graver Water Conditioning Co. division of Union Tank Car Co., New York City, has appointed **Howard G. Egginson** as district sales manager in New England.

Stephen E. Malaker was recently appointed as professor of nuclear engineering at Newark College of Engineering, Newark, N. J. In conjunction with his new appointment, he has relinquished his duties as general manager for Daystrom Nuclear Division, Daystrom Inc., West Cadwell, N. J.

Ralph Reynolds has been appointed sales manager for Acoustica Associated, Mineola, L. I. He was previously technical sales manager for the firm.

Paul V. Malloy has recently been appointed vice president, operations for Kemet Co., division of Union Carbide Corp.

Edward C. Leibig was recently elected president of Corhart Refractories Co. Inc. He will continue as general manager of the refractory division of the parent company, Corning Glass Works.

Hollis G. McLaughlin was recently appointed as manager of industrial engineering for the aluminum division of Kaiser Aluminum & Chemical Corp., Oakland, Calif.

OBITUARIES

Roy H. Glover

An Appreciation by
Clyde E. Weed

It is with regret that I have to record the death, on March 31, 1958, of Roy H. Glover, chairman of the board of the Anaconda Co., with whom I have been associated for many years.

He was born in Goldendale, Wash., July 15, 1890. His father died leaving his son as sole support of his mother at the age of fifteen. He received his education in Goldendale, and later attended school at Portland and earned his law degree at the University of Oregon.

Change of Address and Personals Form

To be sure that you receive your publications and correspondence promptly, please fill in the form below and send it to **The Metallurgical Society of AIME**, 29 W. 39th St., New York 18, N.Y.:

Name _____

Old Address _____

_____

_____

New Address _____

_____

_____

Personals: Please list below your former company and title and your new title and company (or new work) for use in JOURNAL OF METALS. (Copy deadline for personals items is six weeks before date of issue.)

Former Company _____

Former Title _____ Length of Time There _____

New Company _____

New Title _____ Date of Change _____

Any recent activity that would be of interest to members:

_____

_____

_____

_____

PROPOSAL FOR AIME MEMBERSHIP

I consider the following person to be qualified for membership and request that a membership kit be mailed to him:

Name of Prospective Member:

_____

Address

_____

_____

_____

Name of AIME Member:

_____

Address

_____

_____

_____

In 1910, he married Helen Henderson, who survives him. Upon graduation in 1915, Mr. Glover practiced law until he enlisted in the Army. After his overseas duty in the Army, and his discharge in 1919 as a sergeant, he entered the service of the Great Northern Railway Co. in its legal department in Great Falls, Mont., where he combined his duties with the Railway with those of a growing law practice. He was the member of the firm Weir, Clift, Glover & Bennett from 1919 to 1936, and a member of Cooper, Stephenson & Glover from 1936 to 1943. Some of his clients were The Anaconda Co., Montana Power Co., Great Northern Railway Co., and the Chicago, Milwaukee, St. Paul Railway Co.

In 1943, he joined the Anaconda Co. to devote his energies to the production of metals for the war effort. Two years, after moving to Butte, he became western general counsel, and in 1951 Mr. Glover became vice president and general counsel. His rise to the chairmanship in 1935 recorded an advance rare in American industry, for in the space of 12 years he advanced from counsel to chief executive of this corporation.

Mr. Glover brought to the chairmanship of the Anaconda Co. a keen interest in human affairs, and a broad understanding of the importance of nonferrous metals in the economic structure of the free world. During his term of office, the Chilean Government enacted the new Chilean Copper Law which has received wide approbation as a sound basis for the Chilean fiscal system. Recognition of the service rendered as an unofficial ambassador of good will is evidenced by the high honor bestowed April 6, 1955, when president Ibanez awarded him the rank of Knight Commander, Order of Merit of Bernardo O'Higgins, the highest Chilean Decoration which can be bestowed upon a foreigner.

On Feb. 7, 1956 he became an associate member of the Institute of Political and Administrative Sciences, University of Chile, a distinction which identified him with cultural endeavors of Chile. It was the only time in the history of the University that a foreigner was so honored.

On Jan. 2, 1956, Senator Mansfield of Montana announced that Mr. Glover had been named Montana's *Man of the Year*, for 1955. He was chosen from more than a dozen men in the fields of politics, industry, education, and other activities in a poll conducted by the United Press.

In addition to his directorships in the Anaconda Co. and its subsidiaries, Mr. Glover was a director of The First National City Bank of New York and the First Bank Stock Corp. of Minneapolis. He was a member of American and Montana Bar Associations, Federal Power Bar Association, AIME, National Industrial Conference Board, and Academy of Political Science. He held an honorary degree, LL.D., from the Montana School of Mines (1957).

Harrison L. Goodman, (Member 1957) passed away on Mar. 7, 1958. Born in Harriman, Tenn., Aug. 12, 1903, he attended Washington and Lee University, Lexington, Va., in 1921. A year later, Goodman transferred to Cornell University, Ithaca, N. Y., and graduated there in 1926 with an M.E. degree. He returned to Cornell in 1928 and continued with advanced studies in metallurgical engineering until 1937. At the time of his death, Goodman was serving as project engineer for Rust Engineering Co., Birmingham, Ala.

MEMBERSHIP

Proposed for Membership
Metallurgical Society of AIME
Total AIME membership on Feb. 1, 1958 was 29,911; in addition 2,959 Student members were enrolled.

ADMISSIONS COMMITTEE

J. H. Scaff, Chairman; Alfred Bornemann; F. B. Foley; T. D. Jones; L. L. Seigle; J. M. Warde; H. K. Worke.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

Members

Orrin S. Braden, Youngstown, Ohio
Kuang Lu Cheng, Utica, N. Y.
Edward B. Dismukes, Birmingham, Ala.
James E. Dore, Milford, Conn.
Laurence I. Dumpert, Chicago
Willard W. Dunham, Jr., Westfield, N. J.
Thomas A. Evans, Highland, Ind.
David K. Felbeck, Washington, D. C.
Robert J. Fox, Duquesne, Pa.
Lawrence L. Gilbert, Glendora, Calif.
Donald A. Hayes, Gary, Ind.
Charles E. Hicks, New York City
Mats H. Hillert, Stockholm O, Sweden
Joseph S. Iwanski, Bayonne, N. J.
Joseph B. Kearfott, McKeesport, Pa.
W. A. Knepper, Pittsburgh
Emerson N. Ludington, West Haven, Conn.
William H. McClure, Sharon, Pa.
George G. Mueller, S. Duquesne, Pa.
James L. Oberg, New York City
Robert H. Rook, Crown Point, Ind.
Francis J. Schmidt, Sheffield, Ala.
Kermit V. Smith, Gary, Ind.
Freeman R. Whiting, Westfield, N. J.
Nicholas S. Zacklan, Centerline, Mich.

Associate Members

Jay G. Bumpers, Granite City, Ill.
Earl Flodin, Chicago Heights, Ill.
Malcolm R. Harris, Cranford, N. J.
R. Gerald Hay, Chicago
Otto A. Jaros, Chicago
E. E. Kroner, Chicago
Charles J. Nock, Jr., Cleveland
James R. O'Connell, Chicago
William J. Power, Chicago
James J. Sheeren, Chicago
Emmett A. Torney, San Francisco, Calif.

Junior Members

Sumner E. Braumbaugh, Chicago
William M. Halford, Philadelphia, Pa.
Martin D. Harvey, Ferndale, Mich.
Lorin L. Johnson, Salt Lake City
Edward S. Pan, Newark, N. J.
Richard S. Rein, Youngstown, Ohio
Donald H. Shaw, Indianapolis, Ind.

REINSTATEMENT TO MEMBER

Wilfred J. Cooke, Port Colborne, Ont., Can.
Walter W. Edens, Wauwatosa, Wisc.

REINSTATEMENTS—CHANGE OF STATUS

Junior to Member

George Aggen, Natrona Heights, Pa.
William F. Hergert, Columbus, Ohio
Richard E. Wayman, Hammond, Ind.

Student to Member

Bruce W. Dunnington, Columbus, Ohio

CHANGE OF STATUS

Associate to Member

John D. Alstetter, Dearborn, Mich.
Calvin R. Cupp, Deep River, Ont., Can.
Saegusa Fumihiko, Katahiracho, Sendai, Japan
Frederick R. Lorenz, Jr., Pittsburgh
William H. Magruder, Ashtabula, Ohio
Adolph E. Palty, Utica, New York
John J. Skwarlo, Crown Point, Ind.

NECROLOGY

Date Elected	Name	Date of Death
1928	Alexander Hamilton Bell	Feb. 24, 1958
1953	Lehman E. Broxman	Dec. 23, 1957
1951	Rupert Garrison	May 12, 1957
1948	Roy H. Glover	Mar. 31, 1958
1957	Harrison L. Goodman	Mar. 7, 1958
1916	B. W. Knowles	Nov. 14, 1957
1954	Phillip Maverick, Jr.	Nov. 29, 1957
1953	Leroy A. Palmer	Unknown
1947	Edward C. Schwartz	Feb. 1, 1957
1950	E. H. Stevens	Mar. 26, 1958
1957	O. C. Wilson	Unknown

PROFESSIONAL SERVICES

Limited to AIME members, or to companies that have at least one AIME member on their staffs. Rates $50 per year per inch.

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PATENT MATTERS handled . . . TECHNICAL TEXT translated from Japanese into English or vice versa: 2¢ an English word.
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COMING EVENTS

June 22-29, American Society for Testing Materials, Annual Meeting and Exhibit, Hotel Statler, Boston.

Oct. 3-4, AIME NOHC Southern Ohio Section, fall meeting, Deshler-Hilton Hotel, Columbus, Ohio.

Oct. 16, AIME Utah Section, *Progress in Air Pollution Control*, Salt Lake City.

Oct. 27-30, AIME Fall Meeting of The Metallurgical Society, sponsored by the Institute of Metals Division, Carter Hotel, Cleveland.

Nov. 7, AIME Pittsburgh Section, Thirteenth Annual Off-the-Record meeting, Pittsburgh.

Nov. 17-20, The Fourth Conference on Magnetism and Magnetic Materials, sponsored by the American Inst. of Electric Engineers, in cooperation with The American Physical Soc., the Inst. of Radio Engineers, The Metallurgical Soc. of AIME, and the Office of Naval Research, Sheraton Hotel, Philadelphia.

Dec. 3-5, AIME 16th Electric Furnace Conference, Statler Hotel, Detroit.

Dec. 11, AIME Utah Section, Panel discussion on the Place of Research in the Minerals Industries, Salt Lake City.

Feb. 15-19, 1959, AIME Annual Meeting, St. Francis Hotel, Sheraton-Palace Hotel, and Sir Francis Drake Hotel, San Francisco.

Apr. 2-3, 1959, AIME Technical Conference on Stress Corrosion, Mellon Institute, Pittsburgh.

Apr. 6-8, 1959, AIME 42nd National Open Hearth Steel Conference and Blast Furnace, Coke Oven, and Raw Materials Conference, Jefferson Hotel, St. Louis.

Apr. 27-30, 1959, AIME International Symposium on the Physical Chemistry of Process Metallurgy, Penn-Sheraton Hotel, Pittsburgh.

June 13-15, 1960, first International Powder Metallurgy Conference, Hotel Biltmore, New York, under joint auspices of AIME and MPA. Fritz Lenel, RPI, representing AIME, and Kempton Roll, MPA, will be co-chairmen of the Conference.

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J. M. Mathes, Inc.

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Frankel Co., Inc. ..... Second Cover
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Marschalk & Pratt

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The Griswold-Eshleman Co.

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Ramtite Co., The ..... *
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The Aitkin-Kynett Co.

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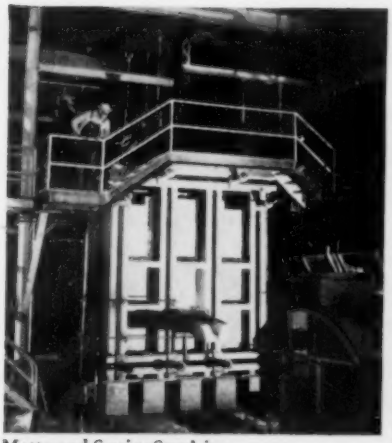
* Previous issue.



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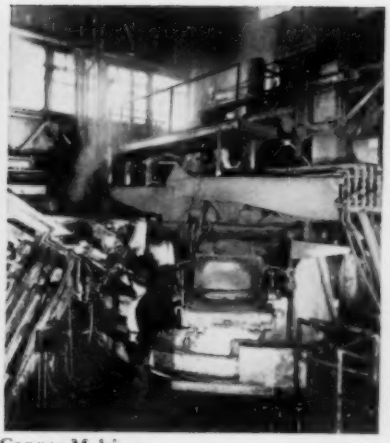

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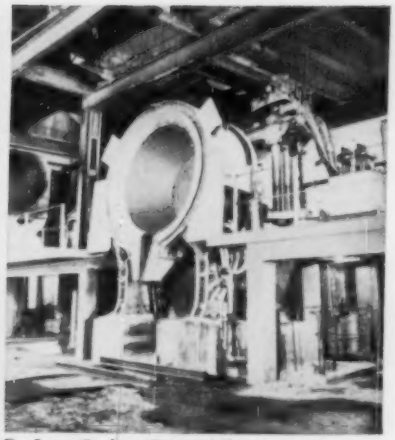

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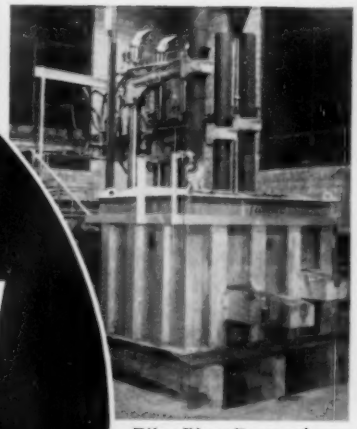

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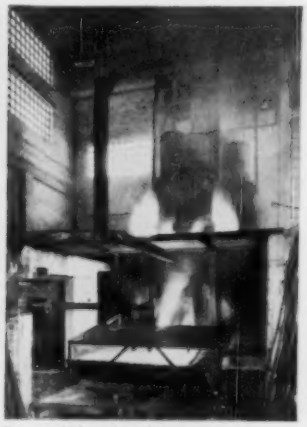

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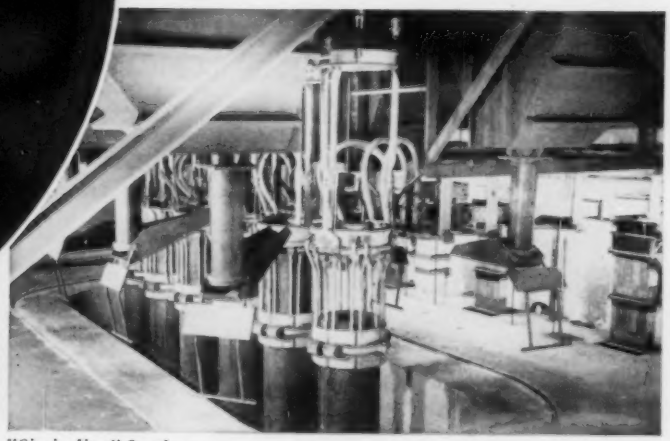

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